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PREFACE

This volume is the third volume published in 1916. It contains the papers and discussions relating to iron and steel and allied subjects presented at the San Francisco Meeting of September, 1915, and the New York Meeting of February, 1916.

Vol. LIV will contain the rest of the papers and discussions presented at the New York Meeting, February, 1916.

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PAPERS

PAPERS

The Geology of the Iron-Ore Deposits In and Near Daiquiri, Cuba*

BY JAMES F. KEMP,† NEW YORK, N. Y.

(San Francisco Meeting, September, 1915)

THE iron ores of southeastern Cuba present a subject of exceptional geological interest. Their relations to the inclosing rocks are varied and in some cases unusual. The problem of their geological origin is not a simple one and for this reason has occasioned some divergence of views among previous observers. Profiting, however, by these earlier studies and by more detailed field and laboratory observations as well as by the experience of added years of mining, much more satisfactory descriptions and interpretations can now be prepared.

PREVIOUS WORK

In 1884, interest in the southeastern coast of Cuba became active among the consumers of iron ore along the Atlantic seaboard of the United States. Active development was undertaken by the Juragua Iron Co., Ltd., a corporation affiliated with the Bethlehem Steel Co., and the Pennsylvania Steel Co. The property of the Juragua company, which is situated about 15 miles east of Santiago, was studied in June and July, 1884, by Dr. James P. Kimball, at the time Professor of Economic Geology in Lehigh University and one of the best known of American mining geologists. Dr. Kimball published on his return two papers, the first important descriptions which we have of the local geology.[1] It is

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* In the preparation of the present paper the writer has had the cordial support and interest of Charles F. Rand, President of the Spanish-American Iron Co. For this aid the fullest acknowledgment is here gladly made.

† Professor of Geology, Columbia University.

[1] James P. Kimball. Geological Relations of the Specular Iron-Ores of Santiago de Cuba, *American Journal of Science*, vol. xxviii, third series, pp. 416 to 429 (1884). The Iron Ore Range of the Santiago District of Cuba, *Trans.* xiii, 613 to 634 (1884–85). To the combined reprints of the above papers as distributed to his friends, Dr. Kimball added reproductions of three maps which accompanied his report to his clients. The first is of the East and West Mines near Firmeza, on a scale of about 150 ft. to the inch and with contours at 3-m. intervals. Some notes on the geological formations are written on the map. The second contains three separate figures, viz.: (1) an outline map of the coastal district from the Bay of Santiago to the Rio Bucano; (2) a geological section through the Juragua Hills to the Gran Piedra; (3) a claim map of the surroundings of Firmeza. The last plate contained a series of geological sections and panoramic sketches of the mines and orebodies near Firmeza.

indeed true that 28 years earlier, D. T. Ansted,[2] an eminent English geologist, had recorded an account of the copper deposits of El Cobre, *west* of Santiago, and of others near Cienfuegos. From the earlier contribution one may glean a few details of the local geology at these two mines, but to Dr. Kimball we owe the first easily accessible descriptions of the region to the *east* of Santiago.

Dr. Kimball describes a syenite as the basal and oldest rock along the coast. He saw its outcroppings in a belt near the sea and learned of its supposed reappearance in the higher portions of the mountain range which culminates in the Gran Piedra. He therefore inferred that it existed beneath the diorites which form the country rock of the intervening belt and which contain the iron ores. Near the sea Dr. Kimball observed the coral reefs which obviously had been raised above the water in recent geological time and which stood at three elevations, respectively 14 ft., 175 ft., and 350 ft. above tide. Farther back the diorite was believed to cover the syenite, through which in some places it may be seen coming up in dikes. The diorite was believed to be thinnest toward the coast and to reach a maximum of about 2,000 ft. farther back. Dr. Kimball inferred that the diorite welled up as an outbreak of igneous rock through the syenite and that from sources of supply back toward the Sierra Maestra it flowed southward toward the sea. The flood of igneous rock was either checked by the salt water or failed in amount and so covered no greater area than approximately that which we now observe. In its course the diorite was supposed to overwhelm reefs of coral limestone, perched on the syenite and still older than those now visible. The effect of the diorite was to turn these older reefs into the crystalline marble which may sometimes be seen in association with the orebodies. Subsequently the circulating rain-waters, acting on the diorites, which contain much iron oxide, took the iron oxides into solution and encountering the ancient coral reefs, were relieved of their dissolved burden so as to replace the limestone with the ore. Dr. Kimball observed structures still preserved in the ore which suggested the organic growth of coral.[3]

These observations are of great interest, but so far as the origin of the ore is concerned are chiefly of value in bringing out the probable precipitation of some of the ore, perhaps indeed of most of it, through the

[2] D. T. Ansted. The Copper Lode of Santiago in Cuba, *Quarterly Journal of the Geological Society of London*, vol. xii, pp. 144 to 153 (1856). Extended quotations from this paper are given by Hayes, Vaughan, and Spencer, *Report on a Geological Reconnaissance of Cuba*, pp. 44 to 50 (1901).

On the San Fernando Copper Lodes near Cienfuegos in Cuba, *Idem*, vol. xiii, pp. 240 to 242 (1857).

[3] The writer has also observed similar cases at Sigua and has been reminded of the supposed fossil coral of the pre-Cambrian rocks of Quebec, the Eozoon Canadense, which was an object of so much study in former years, but which is no longer regarded as of organic origin.

agency of limestone. The precipitating limestone was not an old coral reef perched on the syenite, but a member of an important sedimentary series which is probably the oldest visible rock in the region.

The next paper upon the district is by F. F. Chisholm,[4] who specially examined the Berraco group some six years after Dr. Kimball's studies, and who also visited the Juragua company's mines. Mr. Chisholm reviews the relations of the syenite and diorite, and notes the three terraces of recently uplifted coral. With the view, however, that the ore has replaced ancient coral reefs he dissents entirely. Much better exposures had been prepared at Firmeza in the Union mine during six years of work. A vertical body 200 ft. high and of goodly although not definitely stated width had been exposed. Quite naturally Mr. Chisholm found difficulty in believing that it was an old coral reef replaced with ore. The slabs of limestone observed in the diorite near the ore were interpreted as fragments torn off by the diorite in its upward journey and derived from some older formation. The great vertical orebody at the Union mine was inclosed in diorite. It had a wavy band of chloritic clay on the east side of the ore, with small parallel, lenticular streaks of vitreous quartz. Outside the clay was 2 ft. of sand, loosely compacted and colored by manganese, but displaying a sharp contact against the diorite. The other openings at Juragua were similar but presented less clearly marked relations. At Berraco, by way of contrast, no limestone was exposed. Mr. Chisholm finally interpreted the ore as a direct concentration in an igneous dike, which was originally characterized by a high percentage of iron; or else it is a distinct band forming a portion of a larger dike. The source of the ore would therefore be below, and the ore itself would continue below the effects of atmospheric action.

Two years after Mr. Chisholm's visit, an examination of the Sigua group together with visits at Juragua and Daiquiri was made by the late George W. Goetz of Milwaukee, whose observations were communicated to the late Professor Hermann Wedding of Berlin, partly by letter and partly through short articles in the press. Professor Wedding assembled the data in a paper, although not himself personally familiar with the mines.[5] The oldest rock is stated to be the syenite of earlier writers, upon which rests an oolitic, coralline limestone belonging to the Jurassic and of the horizon of Quenstedt's Beta, of the upper White Jura. The limestone is much disturbed and together with the syenite is pierced by outbreaks of diorite, which have changed the limestones neighboring to them into white marble. The red hematite is associated with the diorite in the most intimate manner as in most localities the world

[4] F. F. Chisholm. Iron-ore Beds at the Province of Santiago, Cuba. *Proceedings of the Colorado Scientific Society*, vol. iii, p. 259 (1888–90).

[5] H. Wedding. Die Eisenerze der Insel Cuba, *Stahl und Eisen*, vol. xii, No. 12, pp. 545 to 550 (June 15, 1892).

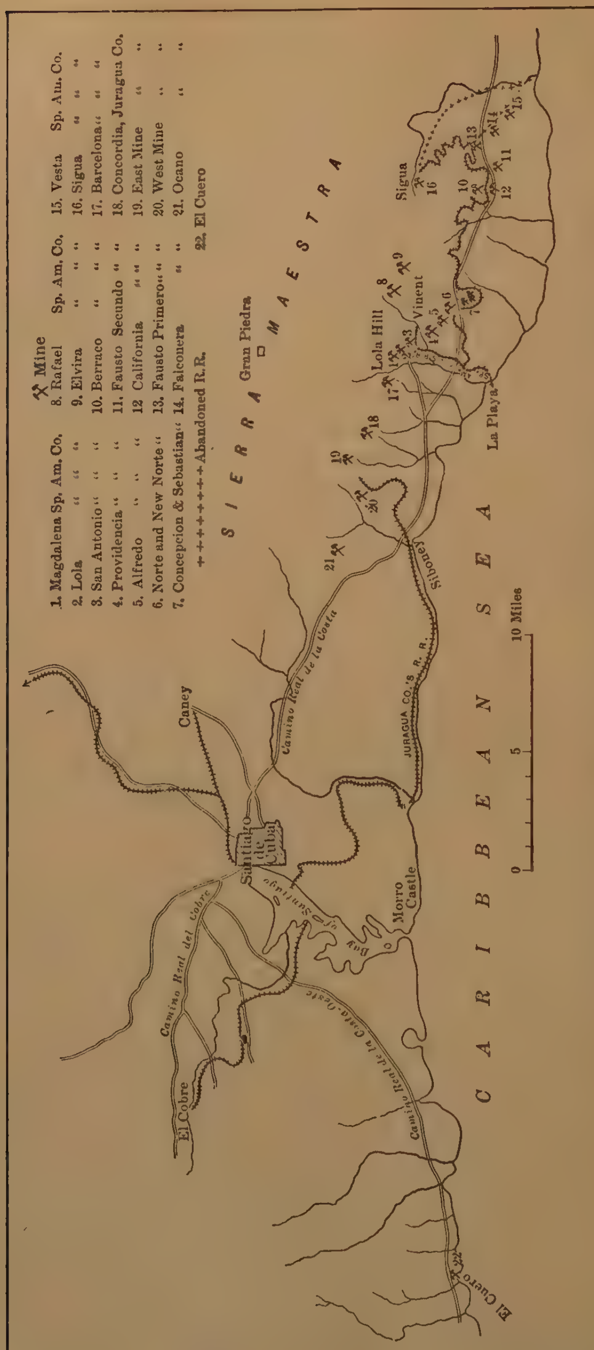

FIG. 1.—SKETCH MAP OF THE MINES EAST AND WEST OF SANTIAGO DE CUBA.

over, such as the Lahn district in Germany, and the Lake Superior district in North America. West from Santiago Devonian sandstone and Silurian limestone constitute the stratified rocks. Dr. Wedding emphasizes the great abundance of surface blocks of ore in the iron districts east of Santiago, and thinks it probable on the analogies with other districts that the iron ores form beds rather than veins. The exposures described as veins may be folded beds. The diorite is often porphyritic and the iron ore is at times involved with chlorite and epidote; and frequently appears simply as a replacement (pseudomorphose) of limestone. Many details of the engineering features follow.

In the interesting work, *Cuba and Porto Rico, with the Other Islands of the West Indies* by Robert T. Hill,[6] a very brief sketch of the geology of Cuba appears on pp. 40 and 41, together with a very small geological map after one by de Castro and Salterain. Limestones of Eocene-Oligocene age constitute the coast east and west of Santiago, with an intrusive mass of granitoid rock at Daiquiri.

The district received careful study from Dr. A. C. Spencer in connection with the reconnaissance of Cuba carried out under General Leonard Wood's direction, during the American occupation in 1901.[7] Dr. Spencer gives an excellent historical sketch of the development of the enterprises and, regarding the engineering features, quotes extensively from a report by Charles M. Dobson, E. M., to the Military Government, a report which must have been made shortly before the date of his own report. Dr. Spencer determined the syenite of earlier writers to be a coarse quartz-diorite, whose exposures are well shown from Siboney to the Playa, the port of the Daiquiri district. It is covered, in the hills near the shore, by a thin veneer of coral rock. The quartz-diorite (sometimes called merely diorite by Dr. Spencer) extends about three miles inland and is succeeded by a belt or zone of intrusive masses of porphyry. "Near the southern edge of this zone the porphyry occurs in extremely irregular dikes and these dikes become more numerous in the higher foothills to the north. It is within these foothills that the iron ore occurs. Above them the slopes of the Sierra Maestra are composed of bedded volcanic rocks with a few intercalated beds of limestone or limestone breccia. The lower flows are of a basaltic nature, but some of the uppermost are rhyolitic, while upon the top of the mountain the massive rocks are overlain by rhyolitic breccias, which constitute the base of a series of volcanic rocks occupying the northern slope of the range. These features are indicated in the accompanying diagram Fig. 2. (Fig. 9 of Spencer) which represents also the general geology

[6] The Century Co., New York (1898).

[7] C. W. Hayes, T. W. Vaughan and A. C. Spencer. *Report on a Geological Reconnaissance of Cuba,* with a geological section from the coast to the summit of the Sierra Maestra and four small sections of the orebodies, pp. 69 to 82 (1901).

and structure of the Sierra Maestra from Cabo Cruz to the vicinity of Guantanamo Bay."

The volcanic rocks are not intimately associated with the iron ores so that the chief interest attaches to the porphyry, which is the main wall rock of the ore. Dr. Spencer was impressed by the association of hornblende schist as well as crystalline limestone with the ore as exposed in the mines of the Juragua Company, but not much in evidence in the workings of the Spanish-American company on Lola Hill. He was led by these observations and by others elsewhere in Cuba to infer the existence of an old metamorphic series, even older than the serpentine so abundant in some parts of the island. The outbreaks of the quartz-diorite and the porphyry preceded the Oliogocene epoch of the Tertiary period, since younger beds on the northern slope of the Sierra Maestra contain fossils of this time. After discussing, at too great length to be cited verbatim here, the derivation of the ores by possible magmatic processes in the igneous mass, and by replacement processes of the included schists and limestones, Dr. Spencer favors the view that the ores were original members of the schist and limestone series and that they have been torn off with varying amounts of their old associated strata and floated upward as included masses in the porphyry. The great slabs or masses of ore are often cracked apart and separated by dikes of porphyry which are regarded as the same rock with the general inclosing walls.[8] Several cross-sections assembled and reproduced here in Figs. 3, 4, 5 and 6, on a reduced scale are given in support of the view. One of

FIG. 2.—DIAGRAMMATIC GEOLOGIC SECTION THROUGH IRON-ORE ZONE FROM THE SOUTH COAST TO THE SIERRA MAESTRA.

1. Diorite of Spencer, called granite in this paper.
2. Diorite porphyry, here called diorite.
3. Iron ore and inclosing schists.
4. Basalt tuffs and limestone breccia.
5. Coarse, acid, volcanic breccia.
6. Coral rock.

(Reproduced from Fig. 9, opposite p. 78, of Report by A. C. Spencer, and slightly reduced.)

[8] In *Trans.*, xlii, 167, 168 (1911), two panoramic views appear accompanying some notes incidental to the Canal Zone excursion.

these with a quotation appears also in J. E. Spurr's *Geology Applied to Mining*, p. 118 (1907) where it is more generally accessible than in the original paper.

In reading Dr. Spencer's description one must bear in mind that the

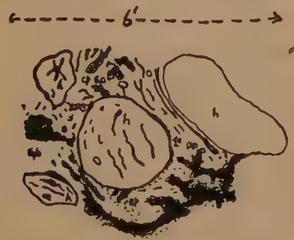

Fig. 3.

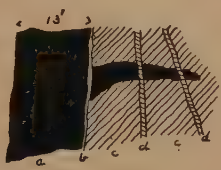

Fig. 4.

Fig. 3.—Magnetite and Epidote near Orebody in East Mine, Firmeza.
h, Hornblende schist; *ep*, granular epidote.
(Reproduced from Fig. 10, p. 80, of Dr. Spencer's Report.)

Fig. 4.—Section through Small Orebody in Northeast Mine of Juragua Company.
a, Magnetite; *b*, selvage impregnated with chalocopyrite; *c*, white marble belonging to the schist series; *d*, fine-grained porphyry.
(Reproduced from Fig. 11, p. 80, of Dr. Spencer's Report.)

rock called syenite by earlier writers is named by him quartz-diorite or simply diorite; whereas the rock called diorite by earlier writers and appearing as the wall rock of the ores is called by him porphyry. Dr. Spencer's quartz-diorite or diorite must not be confounded with the diorite of the others.

Fig. 5.

Fig. 6.

Fig. 5 (Fig. 12, *idem*).—Working Face of Lola Mine, showing General Relations of Orebody to Inclosing Porphyry.

Fig. 6 (Fig. 13, *idem*).—Generalized Vertical Projection of Magdalena Orebody at Daiquiri to Illustrate Intrusion of Igneous Rock.

So far as the sedimentary rocks and especially the limestones are concerned, Dr. Kimball regarded them as coral reefs older than the modern reefs, but although no definite geological epoch is assigned to them, apparently in his view they grew in conditions not greatly changed from those of today except in the matter of subsidence. Professor Hill,

quoting two Spanish geologists, refers them to the Eocene-Oligocene, but Dr. Spencer believes them older than the Oligocene because, on the north side of the Sierra Maestra, Oligocene fossiliferous strata rest upon the same types of eruptives as appear on the south side and are later. Mr. Goetz must have supplied Professor Wedding with some sort of paleontological proof for making so close a determination of them as a subdivision of one portion of the Jurassic. Fossils would be of extreme interest. One cannot well avoid suspecting that they are a continuation of the thick and tilted limestones which one traverses along the wonderful mountain highway, which climbs the escarpment northwest of Santiago to 2,000 ft. above the sea. The winding road crosses in true Alpine manner a great section of well stratified limestones with a northerly dip of 30° or more.

In detailed areal work in the Daiquiri region, as soon as one leaves the artificial exposures of the mines and railway cuts or the freshly eroded ledges of the brooks, the difficulties of the tropics assert themselves. The weathering is so severe and the advance of decomposition so rapid, that fresh rock is often hard to find. Kaolinized and chloritized specimens do not always give a satisfactory indication of the nature of the eruptive rocks. By the aid, however, of E. W. Kohl, Jr., the engineer of the Spanish-American Iron Co., and his assistant, Alfred H. Weaver, a carefully located and quite complete illustrative series of rock specimens, over 800 in number, were collected from the lands of the Spanish-American Iron Co. The general geology has been worked out with the aid of these in the areas not actually visited. Slides for microscopic study have been freely prepared. The writer has also visited the larger mines of the Juragua company and has been further aided by a collection of specimens of ores and rocks made in 1913 by C. R. Corning, to whom acknowledgments are due.

Geographical Résumé

In connection with the iron ores we are deeply concerned with a portion of Cuba which lies east of Santiago along the coast of the Caribbean Sea. The iron mines are limited to a belt which extends not over four or five miles back from the shore. The mines have not as yet been developed more than 20 miles east of Santiago Bay nor nearer than five miles to it. They do not come down to the shore but are first found about a mile and a half from the salt water. They are thus limited to an east and west belt two to three miles wide. The ores are in an exceedingly steep series of hills whose slopes at the larger mines approximate 35°, the angle of repose for loose materials. Some smaller openings are at elevations less than 500 ft. above tide, but the larger ore bodies outcropped originally up to and slightly above 1,000 ft., and cut through

the high, steep hills or ridges, in actual or practically vertical position. The smaller orebodies are of all sorts of shapes and are irregular in their distribution.

To the west of Santiago several enterprises have been started, of which at present only the one at El Cuero, eight miles from Santiago Bay, is active. Its ores lie quite near the shore and are reported as similar in geological relations to those on the east.

Thirty-seven miles westward from the mouth of Santiago Bay is Guama, with a port called Chirivico. The Cuban Steel Ore Co. made extensive developments on some iron-ore lands about 14 years ago, but the enterprise is not now active.

Outline of the Geology at Daiquiri

The oldest geological formation on the lands of the Spanish-American Iron Co., so far as our observations go, is of sedimentary character. Its chief representative today is limestone, but quartzites have also been discovered and at Sigua black slates are to be seen in close association with the limestone. This series is cut up into small irregular exposures by the extensive development of igneous rocks which have greatly metamorphosed it, and have produced contact zones of marked perfection and scientific interest.

The limestone can be seen in the Barcelona claim in a not appreciably metamorphosed condition. It is then a dense blue variety, strongly reminiscent of the Cambrian and Ordovician limestones of the Appalachian area of the United States, and the great Cretaceous limestones of northeastern Mexico. Satisfactory dip and strike were not easy to secure on account of the massive character and the neighboring igneous rock. One reading on the Barcelona claim was N. 55° W., 53° W. The limestone is again abundant to the north of Vinent, and to the northeast toward Sigua. It has been found on the south at various points even so far to the east as the Coco River. While we have no positive evidence that all the exposures belong to the same formation, yet in the absence of the evidence of fossils, no one of which has been yet discovered, we may assume that they do.

Quartzites have been discovered to the east of La Playa, the port of the Daiquiri district, and extend nearly to the Coco River in an east and west belt from a half mile to a mile and a half back from the shore. They are chiefly quartz sandstones of pronounced fragmental character but show the effects of metamorphism. Sericite is recognizable and apparently zoisite. There are particles of feldspar. Fig. 7 will illustrate them.

In the pits at Sigua a fine dark-green slate was found, which under the microscope is a greenish, almost isotropic mass with a few grains of quartz. It was interpreted as a fine feldspathic tuff, now kaolinized and stained green with chlorite.

There are other fragmental rocks both near the mines at Sigua and along the trail to the southeast. They were once feldspathic in composition but are now so greatly kaolinized as not to be more sharply determined. They seemed to be an old volcanic breccia. The strike and dip some two or three miles southeast of Sigua were N. 20° E., 10° W. We have no evidence of the relations of this rock to the limestones and quartzites.

It is possible that in part or as a whole these sediments may represent the ancient metamorphic series mentioned by Dr. Spencer. But no hornblende schist has been found away from the mines, and such ap-

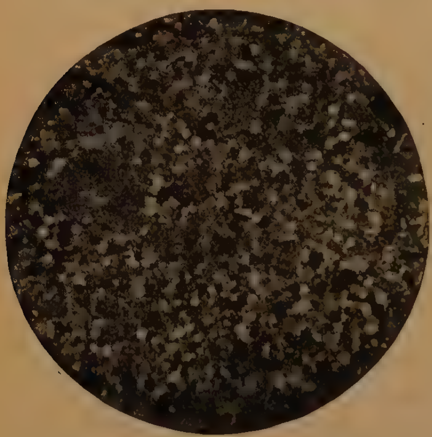

FIG. 7.—QUARTZITE EAST OF LA PLAYA.

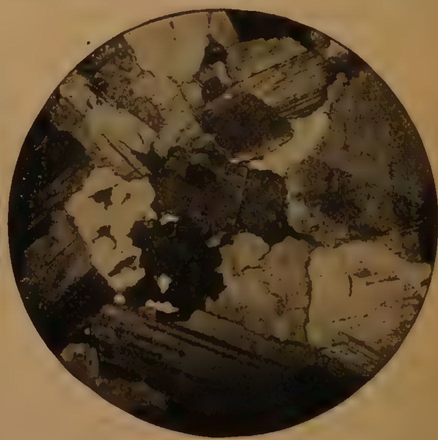

FIG. 8.—GRANITE (QUARTZ DIORITE), BERRACO R.R.

In FIG. 7 and in all the subsequent reproductions of thin sections (except Fig. 43), the actual field is 0.1 in. or 2.5 mm.

parently schistose representatives as were observed in the Spanish-American company's workings were lime-silicates, obviously the results of contact metamorphism. Abundant evidence of faulting movements and of shearing under pressure are to be noted both in the open cuts of Lola Hill and near Firmeza. The limestone-quartzite series does not seem to be so much metamorphosed as would be implied in Dr. Spencer's description of the ancient schists.

Granite.—The review of the earlier papers has shown that there has been a general disposition to regard the syenite or quartz-diorite as the oldest rock in the region. As the writer is anxious to avoid all confusion of this rock with the diorites which are associated with the ore, it will be called granite. The ordinary field worker would unquestionably collect it as granite. When examined under the microscope it always has abundant quartz, making syenite an undesirable name. Sometimes the feldspar is predominantly orthoclase; sometimes, and more often, it is oligoclase. Biotite and hornblende and augite are all three to be seen in

one slide or another but biotite is most common. The usual zircons, titanites and magnetites appear and at times there is a marked tendency to develop micro-pegmatites. The rock is rather coarsely granitoid in texture and of light or actually white color. Dr. Spencer's description of the rock as quartz-diorite coincides perfectly with most of the thin sections examined by the writer. These are, however, phases well over toward granite, which name, both from ordinary field usage and to avoid confusion, is here employed. We know the granite to be later than the limestones because it changes the latter to beautiful white marble and in places has produced contact zones of great perfection. Along the Berraco R. R., where it skirts the east side of the San Sebastian claim there is one contact zone of coarsely crystalline epidote right between the granite and the white marble. The ore of the San Sebastian is in a garnet zone produced by the granite from the limestone; while that of the neighboring Concepcion claim is in epidote produced in the same way. There can be no question that the granite is intrusive in the limestones and is later in age. But the granite itself is at times abundantly provided with angular inclusions of a much darker diorite. In the cuts along the Berraco railway along the east side of the San Sebastian claim, the ledges of granite display a good many of these angular fragments and lead one to believe in some older unidentified member of the composition of diorite. On the other hand, and as all the previous observers have mentioned, dikes of diorite cut across the granite and are generally believed to be the supply conduits for the great masses of diorite which contain the ore. Yet there are dikes in the mines which are also later than the diorite including the ore and which may be of the same group as the dikes that are to be seen cutting the granite. No positive evidence has been obtained by the writer to show that the granite is older than the ore-bearing diorite. The included fragments which appear in the granite may have been derived from the latter rock, unless the earlier observations are destructive of this view. As will be later shown, granitic, pegmatitic and even dacite-porphyry dikes cut the orebodies in one mine and another. Should they be offshoots from the deeper-seated granite, the latter might even be later than the main orebodies themselves. The included, angular fragments would thus correspond to the older diorite which contains the orebodies.

The granite is best developed east and west of La Playa, from which it extends nearly to Vinent. It is also seen in good exposures along the railway from Firmeza to Siboney. Undoubtedly it extends some miles along the coast, Siboney to La Playa and beyond each.

Diorite.—In the neighborhood of Vinent the granite yields to a rather coarsely crystalline diorite which extends some distance to the north and is widespread east and west. The diorite is sometimes visibly porphyritic from phenocrysts of feldspar and for this reason was probably

called porphyry by Dr. Spencer. On careful microscopic study the writer is unable to draw any very sharp distinctions as against the wall rocks of the orebodies and therefore is disposed to regard this great dioritic intrusive as essentially one geologic unit.

The diorite is a gray rock, which may assume a light color on weathering, but which, in the varieties rich in hornblende may be a very dark gray. In some exposures it is of moderately coarse grain, comparable with an average granite having components up to 0.2 in. or 5 mm.; again it is much finer. The most abundant mineral is plagioclase, well twinned and affording extinctions on either side of the plane of twinning from a few degrees to as high as 25. Varieties in the andesine and labradorite series are thus chiefly represented with an occasional member as basic as bytownite. In the coarser varieties the plagioclase has the usual habit of the components of the granitoid rocks. The crystals are roughly as long as broad and of irregular outline. In the more finely crystalline phases there is a marked tendency of the plagioclase to become rectangular and even at times square, reminding one of the characters of the dike rocks. Yet the specimens were certainly gathered from the large intrusive mass as distinguished from the relatively narrow dikes. The plagioclases are rarely zonal. Untwinned feldspar is only now and then to be seen, so that orthoclase is a minor component. Quartz is not uncommon, but it is only a subordinate member. The feldspar may be four-fifths the slide as a maximum and one-half as a minimum. The hornblende is the common green variety in almost all the slides, but it shows frequent indications of having once been brown and to have bleached in alteration. One constantly suspects common green hornblende of having been derived from augite, but the probabilities are very strong that in the Daiquiri rocks the hornblende is an original mineral. In the coarser diorites it is irregular in outline but in the finer-grained varieties in which the plagioclase becomes rectangular, the hornblende exhibits excellent prismatic development. Hexagonal sections and elongated ones with parallel sides are the rule. The hornblende may make up as much as one-third the section; it may sink to a fifth. Irregular magnetite is freely associated with the hornblende, but seldom appears in the feldspar. Apatite is often seen and is even included in the magnetite, being thus an older mineral. Pyrite is now and then an associate of the magnetite.

In some slides and even those in which alteration has not made particularly great progress, calcite may nevertheless be detected. It is secondary and has undoubtedly been yielded by hornblende and feldspar. From this outline, which may be considered the normal, variations occasionally appear. Biotite in a visibly bleached or discolored form is not infrequent as a minor associate of hornblende, but it has never been observed in more than a subordinate rôle. In an abandoned cut in the southern workings of the old El Norte mine a specimen of diorite was

gathered which has much colorless augite associated with the hornblende. The plagioclase at the same time becomes basic. Its extinctions reach 25 degrees with the twinning plane. Epidote has also begun to develop.

FIG. 9.

FIG. 10.

FIG. 9.—DIORITE, LOWER TUNNEL, EL NORTE MINE.

The mineral with parallel bands is plagioclase. The clear white is quartz. The dark mineral is chiefly hornblende with a little magnetite. Crossed nicols.

FIG. 10.—DIORITE FROM EAST WALL OF EAST MINE, FIRMEZA.

The slide is chiefly plagioclase. Hornblende is recognizable in the upper middle part by its cleavage. The very dark minerals on the right lower side are extinguished plagioclase. The rock is practically identical with the one illustrated in Fig. 9.

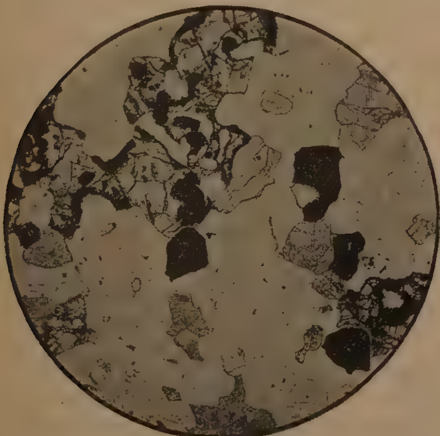

FIG. 11.

FIG. 12.

FIG. 11.—OLIVINE GABBRO IN WHITE LIGHT.

The mineral with high relief is olivine with dark, included magnetite. The gray mineral of less relief is augite. The white mineral is plagioclase. The black mineral is magnetite.

FIG. 12.—THE SAME AS FIG. 11, BUT TAKEN WITH CROSSED NICOLS.

The rock was thus half diorite, half gabbro in its characters, but it seemed to be only a variation from the usual diorite and not a separate intrusion.

Olivine-Gabbro.—A still more marked departure from the usual diorite is found on the Santa Rosa claim, on the southwest extension of Providencia Hill. Excellent fresh rock was gathered on the surface in an area of noticeable but moderate magnetic attraction. When the rock was examined microscopically it proved to be a typical olivine-gabbro, Figs. 11 and 12. Plagioclase, pale-green augite, pale-green olivine, and unusually coarse bits of magnetite are its components. The magnetite develops curious skeleton growths. A very little brownish-green hornblende appears in parallel growths with the augite. It may be that the gabbro is a separate and distinct mass from the ordinary diorite which contains on the north side of Providencia Hill the new Providencia mine and on the south side the Alfredo. Forest growth and concealed exposures made the close tracing of relationships difficult, if at all practicable. The rocks also are not easy to identify sharply without constant appeal to the microscope. Olivine-gabbro makes one apprehensive of titanium in any ore appearing where it constitutes the wall rock.

Wall Rocks of the Juragua Mines.—The same diorite as has been described above from the Spanish-American company's lands appears as the chief wall rock of the Juragua company's East mine and West mine. In the slide from the former, as shown in part, in the photomicrograph, Fig. 10, plagioclase, a little orthoclase, hornblende, a little biotite and one small augite crystal together with magnetite and a little pyrite were observed. In the latter, in a more finely crystalline rock, were plagioclase, hornblende, a little quartz and much magnetite.

Inclusions in the Diorite.—North of Vinent along the Daiquiri River and also on the slopes of Providencia Hill, the diorite is richly provided with angular inclusions of the same dense, dark diorite which we find in the granite, as earlier described. The component minerals of the inclusions are markedly well bounded. Rectangular feldspars, prismatic, sharply six-sided hornblendes and bits of magnetite chiefly constitute them. No pronounced contrasts in mineralogy can be detected as against the usual diorite, but there is more hornblende and the texture is different from the coarser varieties. Apparently the inclusions represent some older solidified rock.

In a recent valuable paper on the Boulder Batholith of Montana, Paul Billingsley[9] has remarked the abundance of dark, angular dioritic inclusions in this famous granite. He has suggested that they represent an older and first chilled basic shell, afterward broken up and included in the later arising granite from a new propulsive effort. The contrasts of the inclusions are so great at Daiquiri, especially with the granite, and diorite is such a common rock, the writer leans to considering the fragments as derived from an older mass, broken up by a newer intrusion.

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[9] *Trans.*, li, 42 (1915).

Alteration of Diorite near the Ore.—Near the orebodies the diorites are noticeably altered and become very unsatisfactory for microscopic study. Epidote appears in secondary veinlets and sometimes in larger veins. Next these veins of epidote the diorite becomes dense and greenish gray, passing practically into a felsite. The process generally summed up under the word propylitization has taken place on a large scale. Dark silicates yield chlorite which stains the whole mass of the rock green. Feldspars become muddy and kaolinized and finally a mass of secondary products must be studied in which one traces the faint outlines and structures of once fresh crystals. So constant is this change near the orebodies that one cannot avoid the conviction that thermal waters have been actively circulating.

White Siliceous Rocks.—In all the larger mines, dikes or at least longitudinal belts of white feldspathic and sometimes quartzose rock are visible, at times in close association with the orebodies and on one side. Lee Reifsneider, the superintendent of the mines on Lola Hill, remarked of this light-colored rock that in passing from diorite into it he looked for ore on the far side; but in passing from ore into it, he anticipated barren rock beyond. In the field it was difficult to decide on the nature of these rocks. They seemed in most instances silicified diorite and were considered to be the result of some thermal action associated with the ore formation. This explanation is not impossible. A specimen was gathered in the Lola-San Antonio workings which under the microscope proved to be a shattered quartz vein with some isotropic mineral filling the cracks, Fig. 13.

A second specimen collected in the Magdalena mine is a siliceous dike of granite or quartz-diorite, akin to pegmatites. Quartz, acidic plagioclase, sometimes in micro-pegmatitic intergrowths, and a little epidote made up the slide, Fig. 14. A third specimen from the East mine of the Juragua company is a beautiful case of a micro-pegmatitic granite dike and is illustrated in Fig. 15. A fourth case from the east side of the West mine of the same company appears to be an old sandstone. The first three cases were believed to be in some way associated with the ore formation, since the acidic rock seemed often to form one wall of the ore. The fourth case, if correctly interpreted must be older than the ore. But besides all these instances there is a fifth on the west side of the West mine at Firmeza, where an undoubted granitic or pegmatitic dike with an offshoot cuts, or in earlier exposures did cut, the ore. It is a richly quartzose rock with quartz in acidic plagioclase, so as often to suggest micro-pegmatite, Fig. 16. A little hornblende and magnetite are also discernible. In the small branch of the larger dike, biotite is the dark silicate. Of these rocks, the sandstone, if correctly interpreted, stands by itself. The first three certainly seemed to have some close relationships with the ore. They might conceivably be

later, but the field relations did not suggest it. At the Concordia mine a white, feldspathic rock was also found associated with the ore. Under the microscope it is chiefly a finely crystalline feldspathic ag-

Fig. 13.

Fig. 14.

Fig. 13.—Shattered Quartz Vein, Lola Mine.

Fig. 14.—Very Acidic Quartz Diorite, consisting of Quartz, the most abundant Mineral in Clear, White, Light or Dark Gray Areas; Micropegmatite, the Mottled Areas; and Epidote, the Dark Shreds. Magdalena Mine.

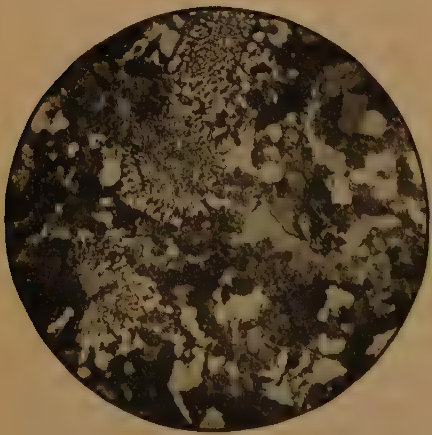

Fig. 15.

Fig. 16.

Fig. 15.—Micro-pegmatite from a Dike in the East Mine, Firmeza.

Fig. 16.—Acidic Dike, West Mine, Firmeza.
Acid plagioclase with micro-pegmatitic quartz on the right. Fine-grained diorite, forming the wall rock on the left.

gregate, reminding one of nothing so much as somewhat badly altered trachytes or bostonite dikes. Blotches of less definite igneous textures appear as well and make the rock very puzzling for sharp determination. Probably the original diorite has been severely affected by thermal

waters, has lost its dark silicates and has changed over to this feldspathic residue.

In the New El Norte mine of the Daiquiri group a light-colored porphyritic dike with branches penetrates the ore in Cut 3 as illustrated

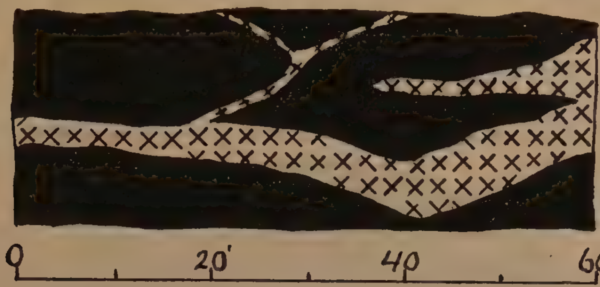

FIG. 17.—RAMIFYING DIKE OR THIN SILL OF DACITE-PORPHYRY CUTTING THE ORE ON CUT 3, NEW NORTE MINE.

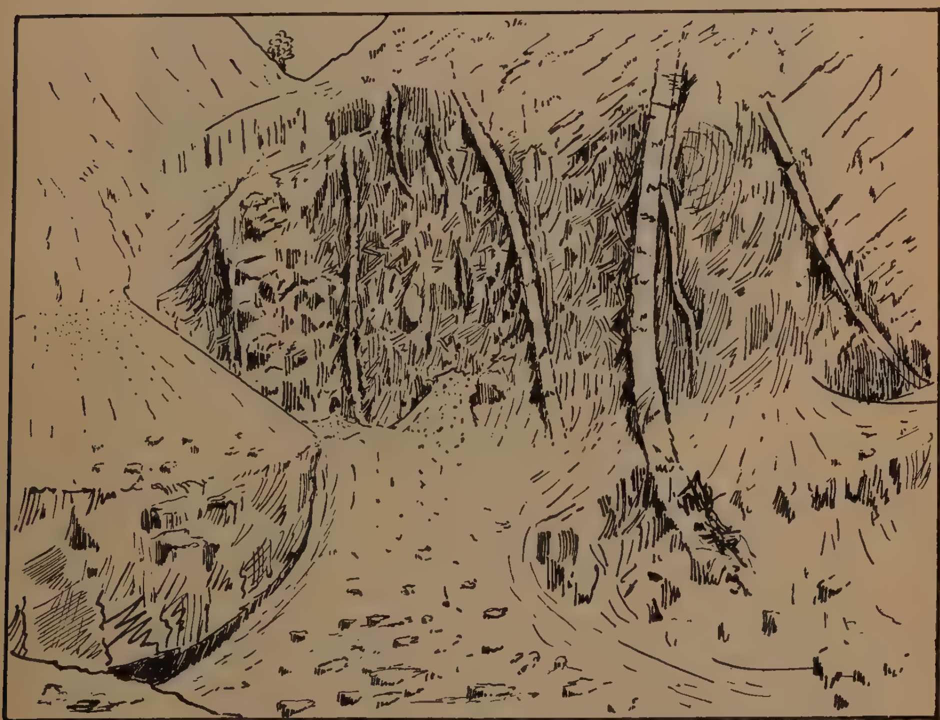

FIG. 18.—NARROW DIORITIC DIKES, CUTTING ORE IN WEST MINE, FIRMEZA. Sketched from a Photograph taken by W. L. Cummings.

in Fig. 17. It is clearly later than the ore and proves under the microscope to be a dacite porphyry. Phenocrysts of plagioclase, rarer orthoclase, and quartz are set in a finely crystalline groundmass of quartz and

feldspar with micrographic texture. Traces of biotite may be determined in the surviving chlorite.

Diorite Dikes.—In the larger mines, such as those on Lola Hill, the main supply of the Spanish-American company, and in the East and West mines of the Juragua company, as well as in some of the smaller openings, narrow black dikes cut wall rocks and ore alike. They are undoubtedly later than both and have entered long after the deposition of the ore. Since the best and freshest ones collected cut also the light-colored feldspathic rock at the West mine of the Juragua company, the

Fig. 19.—Fine-grained Diorite from a Narrow Dike, West Mine, Firmeza, and Showing Characteristic Texture.
The rods and squares are plagioclase, which constitutes the larger part of the rock. The dark minerals are hornblende and magnetite.

narrow dikes are believed to be later than the white rocks. There seem to be two sets of dark dikes. In Stope 6 of the underground work in the Magdalena mine a later one crosses an earlier one. Six from the Lola Hill have been examined in thin section.

One variety is a very feldspathic diorite porphyry which cuts the ore of the Magdalena mine. Phenocrysts of zonal plagioclase are set in a groundmass of smaller plagioclases, all pretty sharply bounded by their own faces. The phenocrysts reach 2.5 mm. but the groundmass is made up of components from a quarter to a tenth this size. A few six-sided

chlorite pseudomorphs after hornblende prisms remain. A little magnetite is dotted through and there is some secondary calcite.

The narrower dikes from Lola Hill have much smaller components of which plagioclase is the chief. The plagioclase shows a marked tendency to develop rectangular or square cross-sections. It is a basic variety. The dark silicates are more abundant than in the diorite porphyry just mentioned. Presumably the common one was hornblende but only chlorite survives to indicate it. The dikes have much fine magnetite. Calcite and epidote are also often in the slides.

An unusually fresh dike from the West mine, Juragua, was 1 ft. thick. It contained, as shown in Fig. 19, beautifully fresh plagioclase which affords rectangular sections. Irregular bits and shreds of leek-green hornblende in less amount than the plagioclases are scattered through the slide, as are fine bits of magnetite. The slide reminds one strongly of diabases, with their lath-shaped plagioclases and irregular augites, but there seems no good reason to regard the hornblende as secondary after augite.

These dark, basic dikes undoubtedly represent the last igneous rocks to enter. They are closely akin to the chief wall rocks of the ore. They are sometimes associated with copper minerals along their borders, so that green copper stains now appear near them and in mining the iron ore it has been possible to throw to one side a few tons, rich enough in copper to send to El Cobre near Santiago. Usually, however, only a few green stains are the result.

Summary of Rocks Associated with the Ore

The general associates of the ore are now before us. They constitute a complex and in some respects a very puzzling group to arrange in a logical series and to place in their time relations. One cannot help associating the granitic or pegmatitic dikes with some large parent body. The natural one is the intrusive granite mentioned at the outset. Yet this granite has produced contact zones on the older limestone with orebodies, whereas the granitic and quartz-porphyry dikes are, in two cases at least, later than the large orebodies. We can only suspect the possible connection without being able to prove it. If true, it would move the Playa granite into a place later in time than the main ore deposition or at least place its expiring effects and offshoots at this late period.

The great mass of diorite certainly preceded the main ore formation, unless one concludes with Dr. Spencer that the masses of ore have been torn off from older formations below and floated upward to their present positions. Details of the ore now to be given cast much doubt upon this hypothesis. Finally many basic dikes and some acidic ones followed the ore formation.

The Ore

The ore is mainly a fine-grained magnetite. Dr. Spencer noted this character although almost all the earlier descriptions speak of it as specular hematite. A belief prevails at the mines that it is a fine aggregate of both

Fig. 20.

Fig. 21.

Fig. 20.—Polished Slab of Ore from Lola Hill, by Directly Reflected Light. The light-gray areas, showing the structure of lath-shaped crystals of plagioclase, are magnetite. The irregular dark patches are quartz and garnet.

Fig. 21.—Nearly the same View as Fig. 20, but with Oblique Illumination so as to bring out the Quartz in the Depressions.

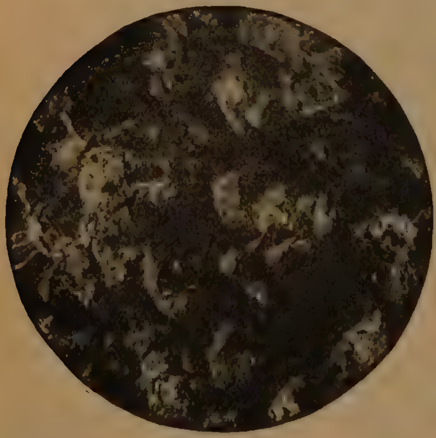

Fig. 22.

Fig. 23.

Fig. 22.—Another Area with Pyrite Standing Out in Relief from the Magnetite.

Fig. 23.—A Still Different Area with Pyrite in Relief.

specular hematite and magnetite, and that analysis has shown this to be the case. Some light might be thrown upon the question by determining the ferrous iron and its relations to the ferric, but care would of necessity

be exercised against pyrite which is widespread in small amounts. A representative suite of specimens taken at intervals from one end of the Lola Hill mines to the other has been tried with the magnet. All but one or two were immediately attracted by it. One or two were inert. We are forced to conclude that the greater part of the ore is magnetite. In the contact zones, as represented in the smaller mines, rosettes of specular ore are not infrequently to be seen.

In a polished slab of ore, illustrated in Figs. 20 to 23 inclusive, the structure is a cellular one with quartz, garnet and pyrite in the interstices of the magnetite. The magnetite itself looks like nothing so much

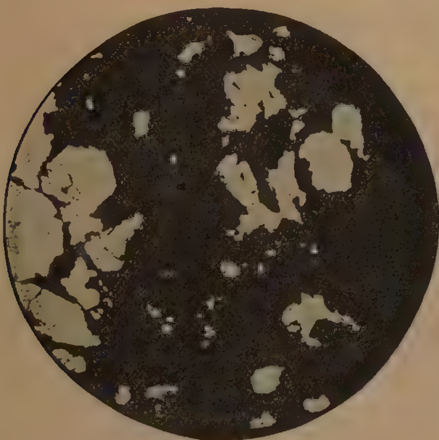

Fig. 24.

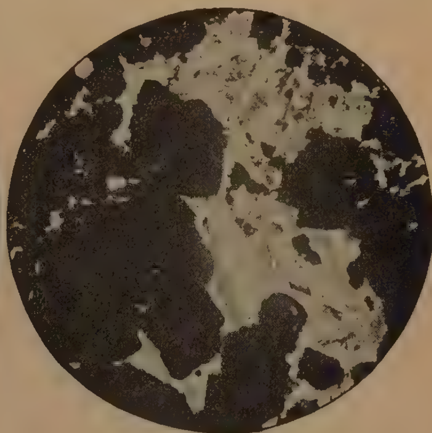

Fig. 25.

Fig. 24.—Quartz Intergrown with Magnetite. Magdalena Mine.
Fig. 25.—Quartz Intergrown with Magnetite in Ore from Firmeza.

as a replacement of lath-shaped crystals of feldspar such as we often see in the diorite itself. One can hardly resist the impression that in the slab some original rock texture is preserved. This texture is most easily explained as due to a replacement of the diorite with ore.

The ore from the portions of the mines above the water level is open textured and cellular because of the oxidation and removal of pyrite, which in the deeper workings appears in unaltered condition. On Lola Hill in the northern workings in the Magdalená claim one notes little in the way of admixture in the ore except pyrite, which increases with depth. But as we pass to the south and into the Lola and San Antonio workings we find notable and significant amounts of quartz. The quartz does not seem to be a secondary and late introduction into cavities in older magnetite, but an original component of the ore. Blades, minute, irregular, angular lumps and dust of magnetite are contained in the quartz. Some magnetite at least apparently crystallized with it (Fig. 24). Similar relations obtain at Firmeza, as shown in Fig. 25.

To a small extent in the ore of the Magdalena mine, but to an increasing degree as one goes south through the Lola into the San Antonio, garnet becomes a component in the ore. Some specimens are a mere mass of garnet and magnetite in blotchy intergrowth. The garnet may even be the chief mineral. Garnet and quartz may both be intergrown and associated in the ore, as shown in Fig. 26. Garnet and calcite are also associated in the ore, Fig. 28. The relations of the garnet and magnetite are so intimate that they must have been produced by the same general process. In one slide from a specimen of ore from the middle of the orebody on a section in Lola Hill, 1,050 ft. from the north end of the

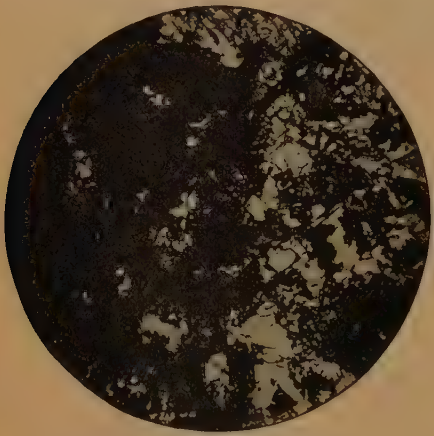

Fig. 26.

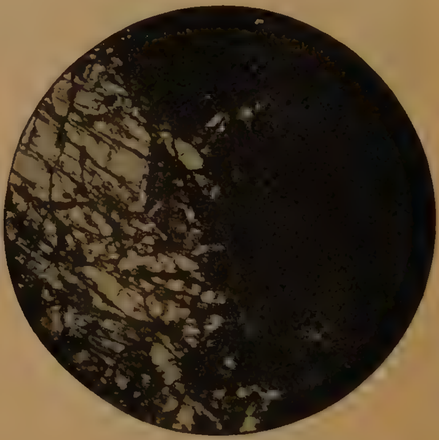

Fig. 27.

Fig. 26.—Quartz and Garnets Intergrown with Magnetite. Magdalena Mine.

Fig. 27.—Olivine Intergrown with Magnetite. Lola Mine.

Magdalena, a brightly refracting, rather high-index, optically positive mineral was found in a slide, which is otherwise chiefly garnet and magnetite. The mineral is irregularly cracked but has also a rectangular cleavage and parallel extinction. It appears to be olivine, although the association is unusual for olivine (Fig. 27). Still one is reminded of the olivine found by Charles Palache and described by C. H. Warren[10] in veins in the Cumberland, R. I., titaniferous magnetite, and obviously an after-effect in an igneous mass. The magnetite at times shows much epidote. The epidote does not seem to be necessarily secondary but to belong with the garnet as one of the original crystallizations in the process of ore deposition.

[10] C. H. Warren. Petrography and Mineralogy of Iron Mine Hill, Cumberland, R. I. *American Journal of Science*, 4th ser., vol. xxv, pp. 35, 36 (1908). An earlier case described from the Alps by E. Weinschenk is cited by Dr. Warren. Weinschenk attributes to it a method of origin such as is here supported for the Daiquiri case.

A sample of garnet was collected in the San Antonio mine and was kindly analyzed by R. B. Herr, Jr., chemist of the Spanish-American Iron Co.

	Per Cent.	Molecular Ratio		Per Cent.	Molecular Ratio
SiO_2	35.23	587	CaO	30.22	537
Al_2O_3	0.82	8	S	0.01	...
Fe_2O_3	30.11	198	P_2O_5	0.05	...
FeO	0.71	10	H_2O +	0.15	...
MnO	0.10	1	$CaCO_3$	2.56	...
MgO	trace	...			
			Total	99.85	

When recast for molecular ratios we have: SiO_2, 587; Fe_2O_3 (198) + Al_2O_3 (8), 206; CaO (537) + FeO (10) + MnO (1), 548. The sesquibases are thus too high, or the silica somewhat low and the protoxides very much too low for an exact garnet formula of 3 SiO_2, $(FeAl)_2O_3$, 3(Ca, Fe, Mn)O, but it is clear that the garnet is predominantly the lime-iron garnet, andradite, most characteristically developed in the contact zones. There was probably some magnetite or hematite in the sample as 2.5 per cent. Fe_2O_3 was soluble in dilute hydrochloric acid.

Along the northeastern side of the San Antonio workings, in the summer of 1914, was an extended streak of garnet rock. Excellent crystallized specimens of a brownish-red variety could at times be obtained where they had projected into a cavity which had later been filled with calcite. This streak or belt of garnet in the diorite seemed attributable to the same general processes as the ore, but it may have been formed from a long included slab or series of slabs of limestone.

Very similar garnet rock may at times be observed in the mines of the Juragua company but the rocks have not been studied in such detail as those at Daiquiri. From the East mine a specimen was gathered which reveals under the microscope pale-green garnet with the ore and quartz mingled. Apparently the quartz was the last mineral to be deposited. In the slide amphibole is sparingly present. From the Concordia mine, garnet rock was gathered which is likewise pale green under the microscope. The garnets are possessed of optical anomalies.

With the ore of some of the smaller mines of the Spanish-American company is associated a black, heavy rock called by the Spanish miners "malo mineral" or "bad ore," and sometimes because of its blue-black color "piedra azul" or blue rock. Specimens were collected both from the New Norte and from the Old Norte mines, and similar occurrences do not fail in other openings. The writer was not always able to distinguish the piedra azul from the ore itself, but on request the miners invariably decided without hesitation. The piedra azul is shown by the microscope to be a phase of the mineralization which led to the ore. The commonest variety is a very fine-grained aggregate of magnetite and

augite, Fig. 29. One would at first take it to be a narrow dike, but in another piece a fine-grained mixture of garnet and pyroxene made up the rock, allying it with the contact effects. The best explanation is therefore that while the mineralizing agents in one place produced ore by replacement of the diorite with magnetite, they may have afforded dense and closely crystalline bodies of magnetite and augite or both these and garnet, close alongside.

One other striking feature of the larger mines of the Spanish-American company should be emphasized before passing to the smaller openings which are based on characteristic contact deposits between intrusives and

Fig. 28.

Fig. 29.

Fig. 28.—Calcite in Magnetite from one of the Mines on Lola Hill.
Fig. 29.—Piedra Azul or Malo Mineral, a Fine-Grained Intergrowth of Magnetite and Augite.

limestones. One cannot study the great open cuts in Lola Hill without being impressed with the evidences of movement and crushing. The diorite is crushed along planes which run in close parallelism with the ore. It is squeezed and sheared around the buttress-like masses of magnetite which have presented a resisting front to the movement, but which probably antedate some though not all of it. The alteration of the diorite is excessive and it has apparently been subject to some influence, especially in the vicinity of the ore which has almost obliterated the hornblende and feldspar in favor of chloritic greenstone. In this crushed zone, traversing Lola Hill like a backbone, the great slab-like masses of ore are distributed along a northwest and southeast line.

Dr. Spencer sketched several cross-sections of the orebodies when at the mines in 1901, which are reproduced in Figs. 3 to 6, along with one from Firmeza. To those in Figs. 3 to 6, is added one more, Fig. 30, reproduced from panoramic photographs of Lola Hill made by the photog-

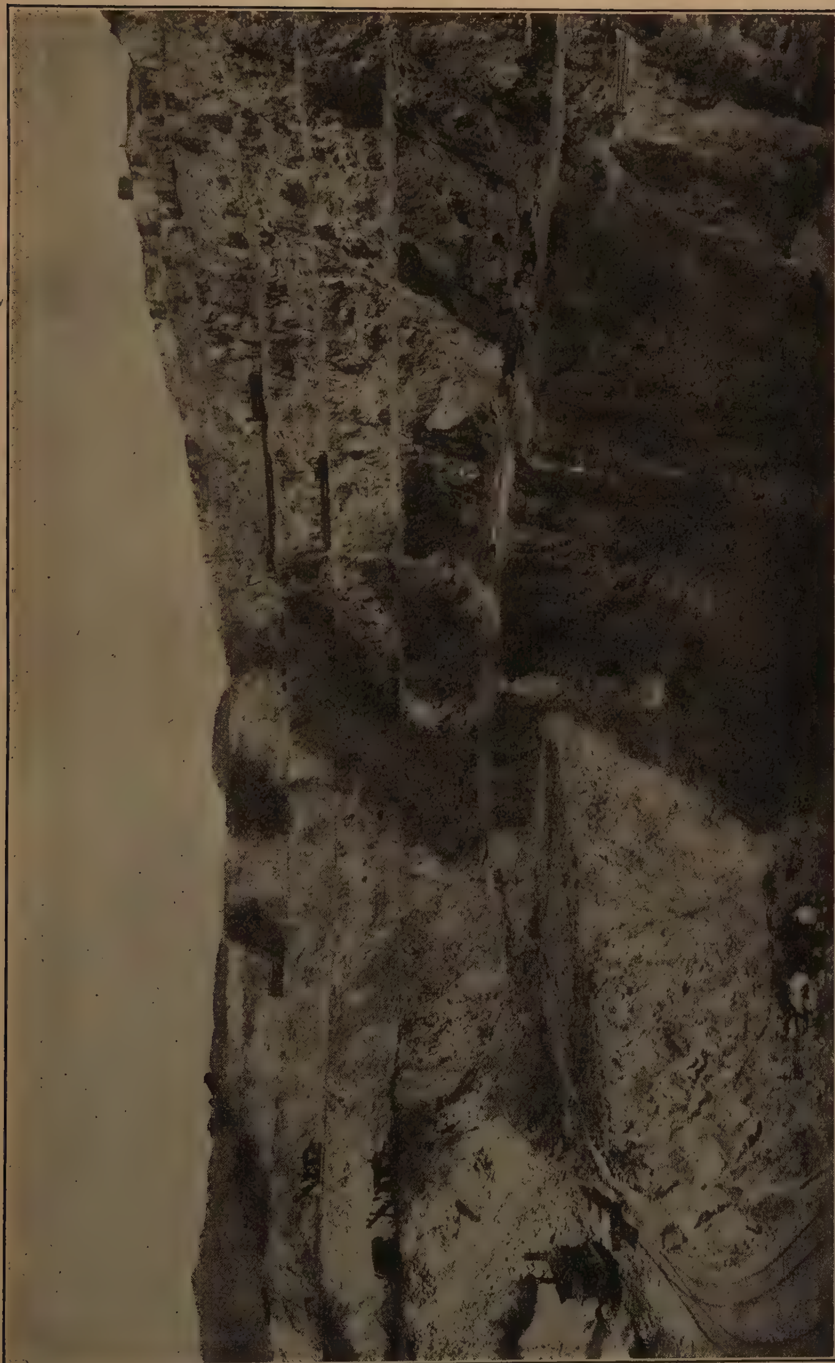

FIG. 30.—PART OF A PANORAMIC VIEW OF LOLA HILL, REPRODUCED TO ILLUSTRATE THE HUGE, TABULAR MASSES OF ORE.

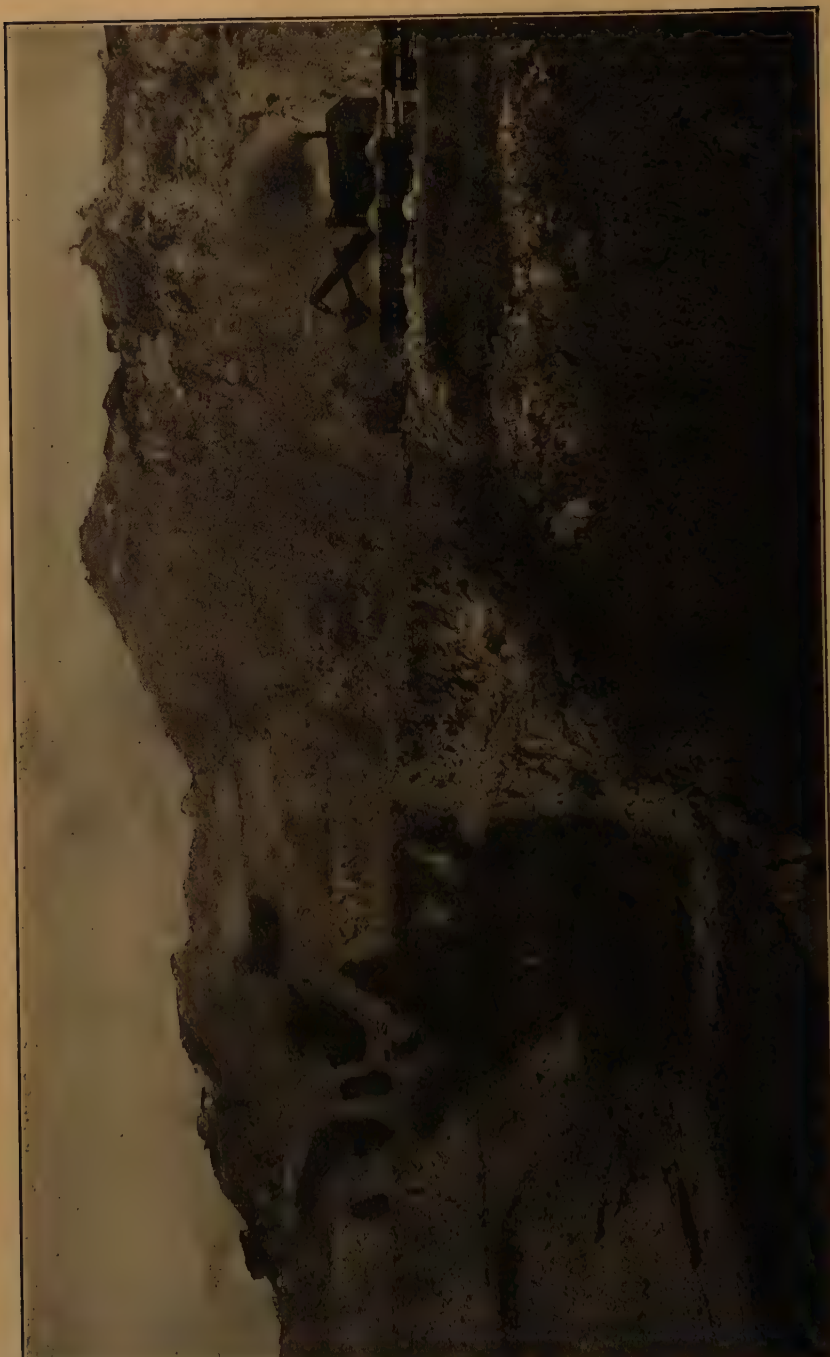

Fig. 30a.—A Second Panorama of Lola Hill taken at a Different Stage in the Operations and Showing a Different Distribution of Orebodies from Fig. 30.

rapher of the Spanish-American company, Francisco Crego. The sketches only afford cross-sections, but we can justifiably imagine that they continue for relatively long distances at right angles to these cross-sections. They are much deeper than wide and in the aggregate, so far as mining has shown, they are on Lola Hill longer than deep. The ore has been followed for 2,800 ft. from the north extreme of the Magdalena

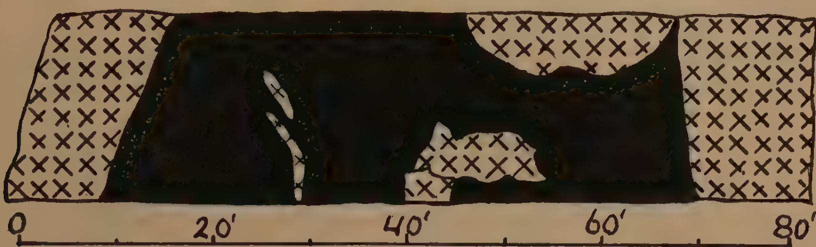

FIG. 31.—SKETCH OF CUT 4, NEW NORTE, TO ILLUSTRATE SURVIVING FRAGMENTS OF DIORITE IN ORE.

mine to the south extreme of the San Antonio. The old summit of the hill with outcropping ore was about 1,000 ft. above tide, while the lowest exploratory tunnel which was cut is 500 or 600 ft. lower.

Thus the whole arrangement of the orebodies is an elongated one, such as would arise from mineralization along a fissured strip. One can hardly imagine such a long and relatively narrow slab or series of slabs to be torn off in depth and floated up edgewise into their present position. On the contrary, the structure of the ore, reproducing the

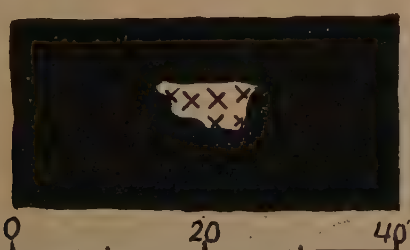

FIG. 32.—MASS OF DIORITE IN ORE, CUT 3, NEW NORTE.
The diorite appeared to be a surviving fragment from a fault breccia.

interlacing feldspar, the associated pyrite, the quartz and the garnets constantly remind us of replacements by just such emissions as we generally believe have produced the contact metamorphic zones.

Another very striking feature is this: When we prolong the general strike of the orebodies on Lola Hill to the southeast across the valley of the Daiquiri River, we run almost, if not quite exactly, into the Providencia mine on Providencia Hill. Along the same general line over

the top and down the other side of the hill is the Alfredo, prolonged again beyond another valley by the Norte and New Norte. They all line up as well as one could expect along a great belt of crushing and faulting. They display much the same characters from end to end, but the greatest mineralization is on Lola Hill.

Effects of crushing do not fail in the smaller mines. Fig. 31 is reproduced from a sketch made in Cut 4 of the New Norte. Fragments of wall rock remained in the mass of ore. In another part of the hill in Cut 3 of the New Norte, relations shown in Fig. 32 were found. A piece of wall rock had survived in the ore along a zone of crushing.

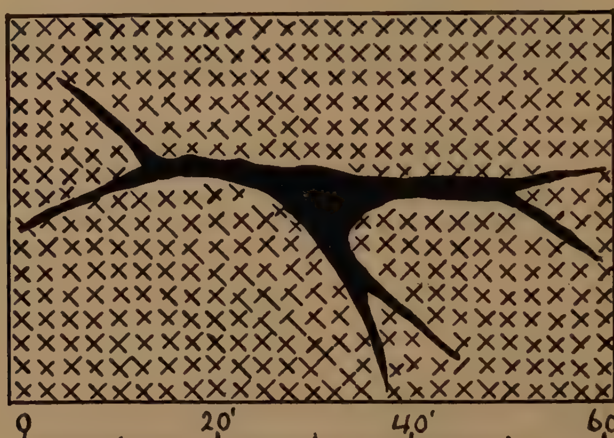

FIG. 33.—RAMIFYING VEIN OF MAGNETITE WHICH APPARENTLY HAD REPLACED THE DIORITE ALONG INTERSECTING FISSURES. ABANDONED WORKINGS OF THE ALFREDO MINE.

One is led to the conclusion that while the diorite mass was still hot in the depths or after it had consolidated and had been penetrated by some other and still hot intrusive in depth, a pronounced northwest and southeast fissured zone was formed, up through which came the emissions, fluid or gaseous, which brought the iron for the ore, the pyrite, the garnet and the epidote; the sulphur for the pyrite; and the silica for the quartz, the garnet and the epidote. The lime required by the garnet and the epidote may have been derived from the plagioclase and hornblende of the diorite, or from included blocks of limestone, or deep-lying limestone, or from several of these sources. The iron probably came up as chloride as has been favored by several writers on contact zones.[11] The relations at Lola Hill are on a large scale not very different from the dike-like masses of magnetite in the intrusive rock of the Iron Springs

[11] C. K. Leith and E. C. Harder. The Iron Ores of the Iron Springs District, Southern Utah, *Bulletin No.* 338, *U. S. Geological Survey*, p. 77 (1908). V. Goldschmidt. Contact-zones in the Christiania District, Norway.

District of Utah, but the largest masses are in contact zones reversing the Daiquiri relationships. E. P. Jennings[12] has considered the dike-like masses at the Utah locality to be intrusive igneous rock. The same conception has been very seriously considered by some observers for Daiquiri although it has not been recorded in print. Nevertheless, the writer favors replacement along a fissured belt as having the strongest claims to confidence (Fig. 33). In drawing conclusions everyone should keep before him all the points made regarding the shape, associations, lineal distribution, microscopic texture and mineralogy of the orebodies.

The Distinctive Contact Zones

There are a number of smaller mines now largely abandoned which are obvious contact zones. They are all in close association with limestone and igneous rock and bear all the marks of having originated by the action of the latter on the former. The two old workings in the Concepcion and San Sebastian are particularly good examples. The former has two pits, a lower on the north side of the hill and not much above the railroad, and an upper and smaller one on top of the hill farther south. The lower cut is entered through granite. A broad zone of epidote succeeds in which were streaks and irregular bodies of ore. The granite had obviously produced from the limestone a contact zone with epidote as almost the only lime-silicate. Together with epidote, bodies of magnetite also resulted. When studied under the microscope the iron ore is also intimately intergrown with garnet and quartz as shown in Fig. 34. On top of the hill the lime-silicate changes to garnet. No obvious reason appeared for the change, but as the two minerals are so closely akin no great difference of physical conditions was probably involved. The Sebastian pit is said to have the ore in garnet also, but darkness prevented the writer from seeing it, on the trip to these claims. On the east side of the Sebastian claim, however, along the Berraco railroad, one passes through cuts of granite when riding south, and then finds a large knob of coarsely crystalline, white marble, extensively changed in places to coarse, prismatic epidote which is illustrated in Fig. 35. The relations are much the same as at the Concepcion lower pit, but no magnetite was noted. No one at all familiar with contact zones produced by intrusive rocks from limestones would hesitate for a moment over the interpretation of these exposures.

The Barcelona claim with two workings of modest size is a case both of ore in the diorite along an apparent vein and in a small contact zone visibly lying between the limestone and the diorite. One walks along

[12] E. P. Jennings. Origin of the Magnetite Iron-Ores of Iron County, Utah, *Trans.*, xxxv, 338 (1905).

Fig. 34.

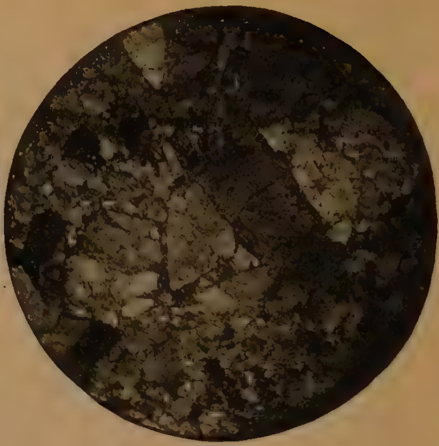

Fig. 35.

Fig. 34.—Relatively Large Garnets, showing Double Refraction and in a Fine-Grained Intergrowth of Quartz and Magnetite. Concepcion Mine.
Fig. 35.—Epidote Rock, from a Contact Zone on the Berraco R.R., produced by Granite from Limestone.

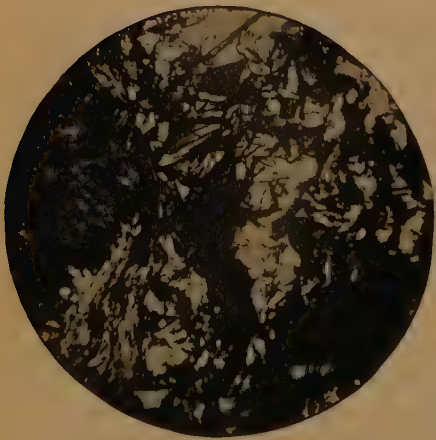

Fig. 36.

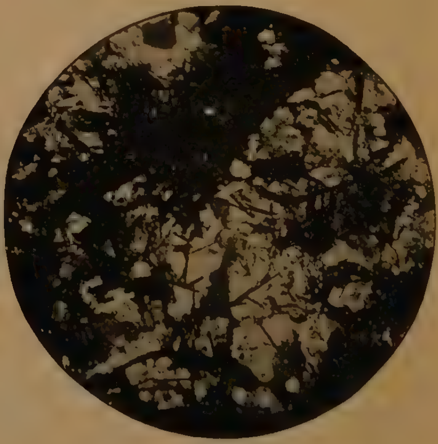

Fig. 37.

Fig. 36.—Platy Growths of Specular Hematite in Quartz. Fausto Primero Mine.
Fig. 37.—Thin Plates of Specular Hematite, branching from Larger Masses of Ore, and Contained in Quartz. Escondido Mine.

an exposure of limestone for 200 or 300 ft. before reaching the small contact zone. No satisfactory dip or strike could be taken, the rock was so broken and disturbed. The limestone was partly white and partly blue, being apparently original blue changed in places to white by the neighboring intrusive.

The Fausto Primero claim on the southeastern portion of the tract is another contact zone. The ore is largely specular hematite in stringers and larger masses in actual limestone. The observer is forced to conclude that the iron-bearing solutions have come in through all manner of crevices and have deposited the specular hematite in irregular networks throughout the rock. Much quartz is mingled with the specularite, which forms platy intergrowths with it, as shown in Fig. 36. In other specimens gathered as a mass of green silicates the microscope revealed quartz and some minute acicular mineral, believed from its small extinction to be tremolite.

Again on the Escondido claim in a small pit, lean ore and a heavy rock, the "piedra azul," were associated. The ore in thin section revealed innumerable plates of specularite set in a quartz mosaic as shown in Fig. 37.

On the Elvira claim a large open cut has been driven so as to win the stringers and irregular masses of ore which are set in garnet and in a fibrous amphibole which is practically coarse asbestos. Asbestos as an associate of the ore was found in abundance on the Rafael claim. The Elvira rock was examined in thin sections and revealed yellow garnet, calcite and the asbestos in very fine radiating fibers, Fig. 38.

The most extensive and the most interesting of the contact zones and attendant iron-ore deposits are at Sigua, high up on a steep and picturesque mountain side looking away across a valley north to the Sierra Maestra where there are coffee plantations still being cultivated on the slopes. All stages of changes can be seen from limestone through the characteristic silicates to ore, but beyond several dikes which cross the zones and are later, no good exposures of igneous rocks were noted. The dikes are a very porphyritic rock with abundant phenocrysts of feldspar, but are too badly decomposed for microscopic study. There are also green chlorite rocks which may have once been more basic intrusives. The contact effects begin with stringers and limited masses of silicates in white limestone. Apparently as at Fausto Primero the ore solutions circulated through all the crevices and open spaces and produced the streaks of minerals by reactions with the limestone. Besides the iron ore, which is largely specularite, well crystallized garnets were produced in the calcite as is shown in Figs. 39 and 40. As the effects grow stronger and the conditions of the largest pit are reached, the silicates become more extensively developed. They may take on finely parallel growths with some surviving limestone and resemble coralline

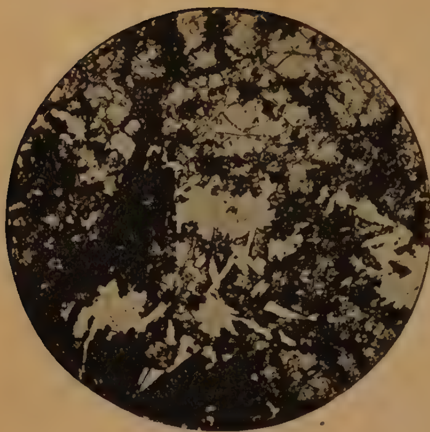

FIG. 38.

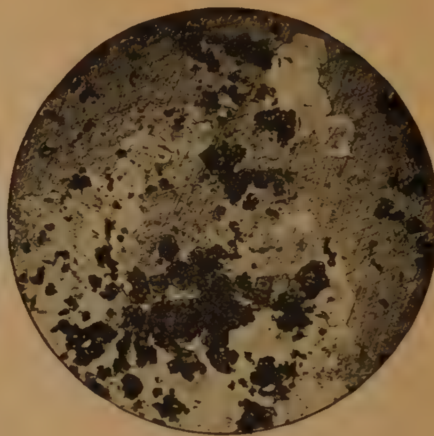

FIG. 39.

FIG. 38.—INTERGROWTHS OF MAGNETITE AND PERHAPS SPECULAR HEMATITE WITH FINELY PRISMATIC AMPHIBOLE AND CALCITE. ELVIRA MINE.
FIG. 39.—RELATIVELY FEW GARNETS (THE DARK MINERALS) WITH QUARTZ (THE CLEAR WHITE MINERALS) IN CALCITE. SIGUA MINES.

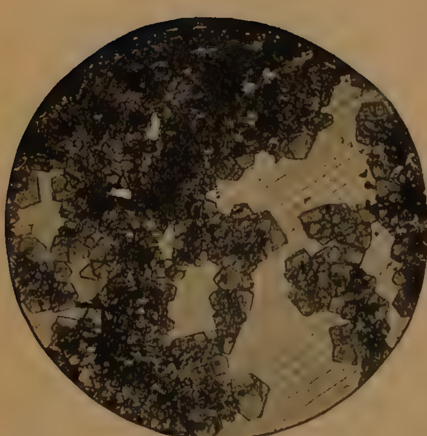

FIG. 40.

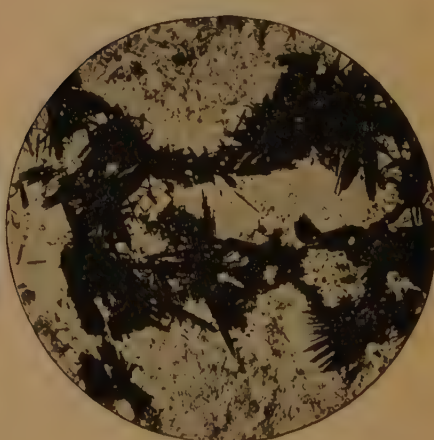

FIG. 41.

FIG. 40.—RHOMBIC DODECAHEDRONS OF GARNET IN CALCITE. SIGUA MINES.
FIG. 41.—PLATES OF SPECULAR HEMATITE IN CALCITE. SIGUA MINES.

structures, strongly reminiscent of Eozoon Canadense. Undoubtedly specimens like these, observed by Dr. J. P. Kimball, in the early days at Firmeza confirmed him in the idea of the replacement of coralline limestone. Sections of the contact silicates reveal all manner of interesting things. We may see yellow garnet, specularite plates, quartz, and diopside all in the same slide. Epidote appears in other slides. The specular hematite in the silicates shows a marked tendency to develop rosettes of radiating plates, Figs. 41 and 42, like roses, as one sometimes sees also in ilmenite. In the slides the plates ramify through calcite more often than through quartz. No reasonable explanation could be given of all these phenomena other than the standard one of contact effects. Some intrusive mass has fed into these limestones gases or

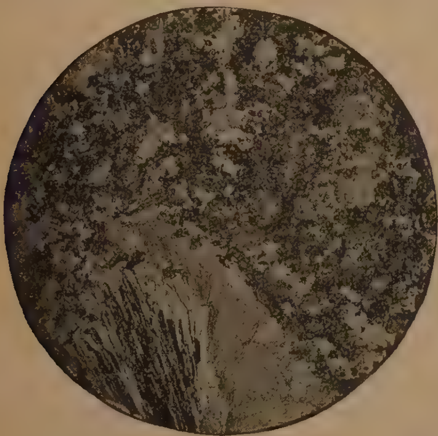

Fig. 42.

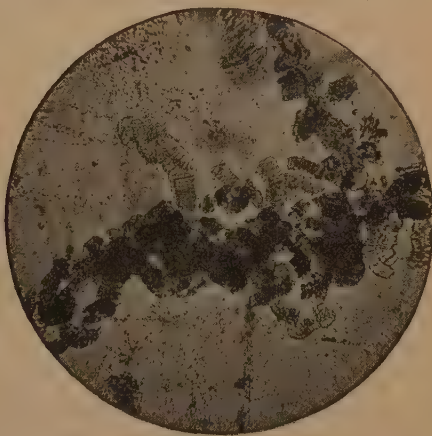

Fig. 43.

Fig. 42.—Thin, Radiating Plates of Specular Hematite in Calcite, below; Finely Granular Epidote in Calcite above. Sigua Mines.
Fig. 43.—Vermicular Growths of Some Undetermined Mineral in Quartz. Actual field, 0.06 in. or 1.5 mm. Sigua Mines.

solutions, or both, which have brought silica-iron compounds, perhaps as chlorides or oxides, and have thereby reorganized the original limestones and produced the lime-iron silicates, the quartz, and the ore. From the mining standpoint the results have led, as in most of the actual contacts in southeastern Cuba, to irregular and seldom large bodies of ore in the midst of barren silicates so that a great quantity of waste must be broken and handled to secure a commercial output. Yet the residual blocks, left on the surface by weathering, originally gave the impression of great bodies underground, and led to the construction of the Sigua standard-gauge railroad to the sea, and to the building of a pier. Floods and the insurrection wrecked the installation after 20,000 to 30,000 tons had been shipped, and since then the Spanish-American company has obtained some 60,000 tons additional by means of a narrow-

gauge extension of the Berraco railroad. Even then 50 per cent. Fe and 12 per cent. SiO_2 limits were maintained. Operations have now been abandoned.

The ore appeared in various small open cuts down in the valley below as well as on the mountain side. There is obviously some complexity to the geology, because in the gulch below the main workings a stratified rock, believed to be an old volcanic tuff and breccia, was observed. In one exposure it had a strike N. 70° W. magnetic, dip 32° N., and again N. 80° W., 20° N. It indicated a marked change in formations from the limestone. In one of the lower pits the black shaly rock was observed, which was earlier mentioned as a probable fine tuff but now altered to a green chloritic mass. On a neighboring claim, the Alfeo, float ore was being picked up off the surface but no excavations were undertaken.

Between Sigua and the other mines much volcanic or at least porphyritic rock appears on the surface. It stretches along the narrow-gauge road for some distance to the south as a green andesite. In thin section the feldspar phenocrysts are plagioclase with chloritic remains of either hornblende or augites which are no longer recognizable. The groundmass is finely micrograniticand contains some minute untwinned mineral which may be in whole or in part orthoclase, albite or quartz, probably the two former. A little magnetite is sprinkled through the groundmass.

To the west of Sigua a purple porphyritic rock was gathered in several places. It proves to be a trachyte. Orthoclase is much more abundant than plagioclase among the phenocrysts, but the groundmass is very finely crystalline, almost felt-like. Alteration is extreme; only limonite remains to suggest dark silicates. The rock was gathered in the general field work and we do not know whether the trachyte appears in dikes or surface flows.

Along the Sigua extension of the Berraco narrow-gauge railroad, and about two miles in a direct line, or three miles by rail southeast of Sigua, two shafts were sunk in former years in exploration of small veins believed to contain silver. Only the dumps and the open vertical holes now remain, but small specimens of barite apparently with tetrahedrite can be picked up. The wall rock is a dense greenstone, evidently derived from some rock like the diorites. No records are available of the assay value of the ore.

Résumé

The iron ores are the subject of chief interest, and in summing up attention will be alone directed to them. They furnish a variety of deposits ranging from those of small size in streaks in limestone associated intimately and microscopically with quartz, garnet, and epidote;

through larger and more extensive developments of the same with the practical extinction of the limestone; to the extreme of ore developed in great tabular masses in diorite, but still associated with the same quartz, garnet, and epidote, both in the large and in the microscopic way. In the large bodies pyrite appears in increasing amount as depth is gained, but it has weathered out of the ore near the surface. The smaller deposits, obviously in the contact zones, have in no case been followed deeply enough to develop much, if any, pyrite and we can only infer its probable presence in depth. Pyrite is quite frequently associated with the magnetites and specular hematites customarily found in contact zones. As regards the magnetite and specularite of the obvious zones, there is little question that they have been yielded by iron-bearing waters or gases from the neighboring igneous rock. So many cases have been studied and so many observers have been led to the same conclusion that little attention is demanded by this type other than the mention of the method of formation. Experience has shown orebodies in contact zones to be irregular in shape and of no uniformity of disposition in the zones. They seem not to be very large in the Daiquiri district, although in the Iron Springs district of Utah and in some other localities in the southwest they are very large.[13] In the larger and more important deposits in the Daiquiri district, such as those in Lola Hill and on Providencia Hill as developed especially in the Alfredo, and for the Old and New Norte, we cannot reasonably go far from a method of origin closely related to that of the contact zones. Their ore with its associated garnet, quartz, and epidote must be closely akin to that of the zones. The long and relatively narrow masses and the obvious location in belts of crushing and rock movement cannot be ignored. The observer is almost irresistibly led to the conclusion that emissions from some cooling, igneous mass beneath, bearing iron and silica, have come up along lines of faulting and crushing. We might justifiably wonder if where we now find the orebodies, slabs or blocks of included limestone chanced to be in the diorite and if they served as precipitants of the iron, so as at the same time to yield the associated garnets. The suggestion is by no means unreasonable for some of the smaller block-like masses, but as one appreciates the extent of the large bodies on Lola Hill, their great length and depth as compared with their width and their uniform vertical or nearly vertical position, the assumption of blocks of limestone is clearly so improbable as not to appeal to reason. The vertical and somewhat straggling line of ore which one sees at Providencia, Alfredo, and at the Old and New Norte could not be explained by it. No ex-

[13] C. K. Leith and E. C. Harder. The Iron Ores of the Iron Springs District, Southern Utah, *Bulletin* No. 338, *U. S. Geological Survey*, p. 73 (1908). A total of 40,000,000 tons is estimated, the larger part in contact deposits, of which one reaches 15,000,000 tons.

planation will fit except one along the lines of vein formation with replacement of the diorite. Where the result was largely magnetite, but so mixed with silicates as to be too low in grade for ore, "piedra azul" resulted. Where the wall rocks yielded more completely to the iron solutions or vapors commercial ore resulted. The diorite was replaced with magnetite and some specular hematite, intermingled with garnet, quartz, and epidote. The lime for the admixed garnet and epidote probably comes from the lime in the plagioclase or in the bisilicates of the diorite. But in the case of the streak of garnet rock in the San Antonio with admixed calcite but no appreciable iron ore, an included slab or series of slabs of limestone is not unreasonable. As stated above, veins of magnetite are known in the diorite porphyry or andesite which has developed the large contact bodies at Iron Springs, Utah. E. P. Jennings has considered them iron dikes, but Leith and Harder regard them as due to emissions from the igneous mass, coming up through cracks in its outer cooled and solidified shell. In some such way as this the Cuban ores have probably been formed.

As was inferred by F. F. Chisholm there seems no reason why the ores should be necessarily limited to relatively shallow depths, but with depth, unoxidized pyrite is quite certain to appear and to contribute relatively high sulphur. Roasting is already practiced at Firmeza.

The larger deposits thus lie along structural lines. The resistance of the ore to weathering processes has left it as the backbones and at the summits of hills whose slopes were mantled with blocks of float. Meantime the original wall rocks have weathered away and have in part been removed by erosion.

The Daiquiri ores are remarkably similar in their geological relations to a deposit near Hong Kong, China, recently described before the Institute by C. M. Weld,[14] and to others in the province of Bulacan in the Philippines as set forth by Dalburg and Pratt,[15] and associated petrographic studies by F. T. Eddingfield.[16] In each case aggregates of preponderating magnetite with specular hematite have resulted by contact metamorphism and replacement. The geological sections in Bulacan might almost be substituted for those in southeastern Cuba.

DISCUSSION

Wallace E. Pratt, Manila, P. I.—I am glad that Professor Kemp has referred to the Philippine ores in his paper, inasmuch as I have had the privilege of examining, in a preliminary way, the ores there. I

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[14] C. M. Weld. *Trans.*, l, 236 to 245 (1914).

[15] F. A. Dalburg and Wallace E. Pratt. The Iron Ores of Bulacan Province, P. I., *Philippine Journal of Science*, vol. ix, No. 3, p. 201 (June, 1914).

[16] F. T. Eddingfield. Microscopic Study of the Bulacan Iron Ores, *Idem*, p. 263.

must say, from the résumé given us, that the Daiquiri ores seem to be similar in origin and character to the magnetite-hematite ores in Bulacan Province, Philippines.

Professor Kemp has also referred to C. M. Weld's description of the magnetite-hematite ores at Hongkong and to the similarity of these ores to the ores at Daiquiri and to those in Bulacan. There is a minor feature of apparent difference in origin in the Hongkong and Bulacan ores. Mr. Weld believes the iron ores at Hongkong to have resulted from the intrusion of the Hongkong granite, a distinctive rock of wide distribution in the Orient, into sedimentaries which are probably of much greater age than those which have been replaced in part by the ores in Bulacan. Associated with some of the ores in Bulacan and elsewhere in the Philippines there is a granite which may be the equivalent of the Hongkong granite; but the Bulacan ores are not related in origin to the intrusion of this granite, which is clearly older than the ores, but to the later intrusion of dikes and small bodies of basic rocks into the granite and into associated tuffs, breccias, and sedimentaries, including limestone. I gather that the intrusives to which the mineralization is related at Daiquiri are also small in volume and later than larger bodies of associated igneous rocks, from which it would appear that the Bulacan ores are even more closely related to the Daiquiri type than are the Hongkong ores.

The deposits in Bulacan are inaccessible and are not large, the largest one containing about 1,000,000 tons, according to an estimate based solely on its outcrop dimensions. Ores of similar character, however, occur in Camarines Province in apparently larger quantity and in a situation on the coast adjacent to deep water.

The Iron Deposits of Daiquiri, Cuba

BY WALDEMAR LINDGREN AND CLYDE P. ROSS, BOSTON, MASS.

(New York Meeting, February, 1916)

Introduction

To the miner, as well as to the geologist, the eastern part of Cuba is a most interesting region. Here we find, in contrast to the moderate relief predominating elsewhere in the island, an imposing mountain range, the Sierra Maestra, extending east and west parallel to the coast, its precipitous front facing the blue Caribbean Sea. In a geological sense this range is largely an unknown land, the only well-explored region being that in the vicinity of Santiago.

The range also contains the most important mineral deposits of the island. They comprise, first, a series of iron deposits yielding a partly hematitized magnetite with low content of phosphorus; second, a remarkable copper-bearing vein at El Cobre; and third, various manganese deposits. Probably more deposits will be found, for the larger and western part of the range is as yet little explored and its slopes are covered by a thick tropical jungle.

The following notes are based on a short visit by the senior author in January and February of 1914 to the mines at Daiquiri, El Cobre, and Mayari. For many courtesies and great assistance he is deeply obliged to Charles F. Rand, President of the Spanish-American Iron Co.; to George W. Pfeiffer, General Manager, and to the several members of his staff.

Geological Features of the Sierra Maestra

For our knowledge of the geology of the Sierra Maestra we are indebted to reconnaissance work by Fernandez de Castro, R. T. Hill, W. C. Hayes, T. W. Vaughan, and A. C. Spencer. The summary by the latter three geologists furnishes the best guide to the region.[1] The range is undoubtedly outlined by a great east-west dislocation. Its southern slope drops abruptly to the sea, while on the north side a much gentler declivity leads down to the rolling plateau of Tertiary limestone which occupies much of the adjacent part of the island. Viewed

[1] C. W. Hayes, T. W. Vaughan, A. C. Spencer: *Report on a Geological Reconnaissance of Cuba*, pp. 69 to 83 (Washington, 1901).

from the sea near Santiago or Daiquiri the summit of the range appears as an undulating line with an elevation of from 2,000 to 3,000 ft. and about 12 km. from the sea. From this crest line an abrupt slope leads down to jungle lands little above the level of the sea, and separated from it by low coast hills, covered by a veneer of coral rock.

The range back of Santiago and Daiquiri seems to end before Guantanamo Bay is reached. At this place the coral limestones of the coast appear to reach far inland, but east of the bay and from here to Cape Maysi the mountains, here dry-looking and covered by brush, rise abruptly from the sea to a level skyline at an elevation of about 2,000 ft., the summit evidently forming a plateau, which probably extends northward, connecting with the high plateaus of Baracoa and Mayari.

Back of Santiago the railroad pass at Cristo is only 560 ft. above the sea while the road crossing the range at Boniato Hill reaches the summit 1,000 ft. above the tide and 12 km. from Santiago. West of Santiago there is a steep and high coast range separated by the Cobre Valley from the lower westward continuation of what may be called the Sierra Boniato. Farther west this coast range becomes much higher, culminating at Tarquino Peak (elevation 8,400 ft.) and dropping off precipitously into the sea.

The geological elements which make up the range near Santiago are as follows:

1. Recent coral rock veneering the coast hills.

2. Oligocene limestones and marls underlying the coral rocks along the low land of the coast.

3. A very thick series of greenstone tuffs and agglomerates dipping northward and well exposed in the Cobre Valley. Age unknown.

4. A thick series of heavy-bedded gray limestone well exposed on the road across the Sierra Boniato north of Santiago. These beds dip north steeply at the base of the ridge, more gently at the summit. Age unknown, but unless faulted they should overlie the tuffs of El Cobre.

5. Andesites, rhyolites, greenstones, and tuffs near Cristo and near the top of the high range back of Daiquiri; probably later than the limestones.

6. A large area of intrusive diorite or diorite porphyry with inclosed fragments of an older, bedded limestone. This diorite is said to form the seaward part of Sierra Cobre and occupies the foot hills and the basal part of the high ridge back of Daiquiri. The iron deposits are found in this diorite. The age of the included limestone and of the diorite is unknown. The idea of the geologists who have done the most work in this section seems to be that the lavas and tuffs and associated limestones are of Eocene age and that they rest on a basement of diorite. Consequently the limestone included in the diorite must be older, possibly Paleozoic. While not able to contradict this statement, we believe that

there is a possibility that the included limestones may be of the same age as those of Sierra Boniato and that consequently the great diorite intrusion might have formed the last feature of the volcanic and the sedimentary period represented by the Cobre and Boniato rocks.

Iron Deposits in the Diorite

As stated above, dioritic rocks occupy a wide area along the coast from the Sierra Cobre to Guantanamo Bay. In the diorite and close to the coast line are found a considerable number of local and limited masses of iron ore which are known to extend from a point 36 miles west of Santiago to Sigua, 25 miles east of that place. There are several mines along this belt of iron deposits; El Cuero mine lies west of Santiago. The Juragua properties owned by the Bethlehem Steel Co. are at Sevilla and Firmeza, from 5 to 10 miles east of the city. The Daiquiri properties owned by the Spanish-American Iron Co. include the Vinent and the Berraco groups, the former comprising the Lola Hill mines described in this paper. The Sigua deposits lie near the coast and still farther east.

The Juragua mines were the earliest opened in this belt and production began in 1884, while the mines at Daiquiri started shipping in 1895. The production from each of these great properties has ranged from 100,000 to 500,000 tons a year. At the time of visit these two properties were the only ones shipping ore. In 1913, the Juragua mines produced 369,213 long tons, the Daiquiri mines 591,322 long tons, and the El Cuero mines 130,183 long tons, a total of 1,090,718 tons.

The total production of the Juragua mines is 6,679,617 tons and that of the Daiquiri mines 6,744,004 tons.[2]

Situation of Daiquiri Mines

The Daiquiri mines are best reached by steamer from Santiago, the trip affording a beautiful view of the shelving coral deposits of the shore line, the intervening low lands, and the imposing front of the range continuous from Sierra Boniato to far east of Daiquiri. At the open roadstead there are a wharf and large ore bins. A railroad extends from the landing place up to the mines, a distance of 4 miles along the Magdalena River, a small water course here flowing through a moderately hilly country. A short distance below the mines the road leaves the river and follows Daiquiri Creek up to the town of the same name, where, at an elevation of 300 ft., the offices of the company are located. The town lies in a basin-like valley, hills rising 500 to 800 ft. above it. Back of these, steep slopes lead up to the main range.

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[2] *Mineral Resources of the United States*, 1913, vol. i, p. 317 (1914).

Interest naturally centers in Lola Hill, which attains a height of 600 ft. above the offices and is located on the ridge separating Magdalena River from Daiquiri Creek, for the principal mines are situated on the top and sides of this eminence. Smaller deposits lie east of the offices and extend irregularly toward Berraco, but little ore is shipped from these places and little definite geological evidence is obtained from them compared to that afforded by the magnificent exposures on Lola Hill.

Geology of Daiquiri

Granite.—A granitic rock with white rounded outcrops forms the most prominent part of the hills at the wharf and extends along the railroad for about 1 or 2 miles inland. The seaward side of the granite hills, which may be 150 ft. high, is covered by recent coral rock in three heavy benches. The granite is somewhat porphyritic and contains abundant phenocrysts of albite, while those of orthoclase, green hornblende, and quartz are less plentiful. The coarse groundmass is allotriomorphic and contains feldspar, quartz, biotite, and small cubes of magnetite. Epidote, chlorite, sericite, and kaolin are secondary products.

Diorite.—The contact between granite and diorite was not carefully studied, but it is probable that the two rocks are not far apart in time of origin.

The diorite continues up to the mines and is the predominant country rock of the district. It is usually light green in color when fresh, weathers in irregular, jagged, whitish outcrops, is traversed by irregular joint systems, and contains in places veinlets and masses of epidote, especially near the iron deposit. Darker varieties form ill-defined masses in the normal rock, which is also often cut by white aplite dikes.

A typical diorite from the east side of the orebody, in the second open cut of the San Antonio mine, is of medium grain and distinctly porphyritic, this being especially noticeable in thin section. Under the microscope the most numerous phenocrysts prove to be of plagioclase and are rarely more than 1 or 2 mm. in length. Many of them are rounded or broken and poorly twinned. The few exact measurements which could be made indicate an oligoclase-andesine. The refraction, accurately determined, is from 1.535 to 1.545, which corresponds well to this mineral.

There are also one or two anhedral, but not corroded, phenocrysts of quartz, as well as several imperfect prisms and granular aggregates of colorless augite.

Sheaves of secondary green hornblende are abundant and needles of the same penetrate the feldspar, all of this hornblende being probably

derived from augite. Small grains of black ilmenite are fairly common, but most of them have been converted to titanite. In another rock of the kind, this process has been carried so far that no black iron ore remains.

The groundmass is microgranular, very interlocking and poikilitic or micro-porphyritic in places. It consists of grains of unstriated feldspar with index of refraction smaller than quartz, intergrown as indicated with quartz.

The secondary constituents comprise, besides hornblende, some epidote and a little sericite, the latter contained in the feldspar; also a little chlorite and calcite.

A partial analysis of this rock, made by C. S. Venable, of the Massachusetts Institute of Technology, yielded 64.07 per cent. SiO_2, 5.62 per cent. CaO, 5.6 per cent. Na_2O, and 0.7 per cent. K_2O. The rock is thus more correctly defined as a normal quartz-augite-diorite-porphyry. The percentages of CaO and Na_2O correspond well with what might be expected and the small amount of K_2O is noteworthy, showing the rock to contain very little orthoclase. The percentage of black iron ore present is no more than would be expected in a rock of this kind.

Aplite.—White aplite dikes of irregular outline are found in places in the diorite. One of these occurs on the ridge between Magdalena River and Daiquiri Creek, ¼ mile north of Lime Point. It is a dirty-white medium-grained rock speckled with little grains of hornblende. The essential minerals are oligoclase, quartz, and orthoclase with a little hornblende, apatite, and titanite. The rock contains numerous feldspar phenocrysts, usually with excellent crystal outlines, and in places the texture of the coarse groundmass may be described as allotriomorphic. Micropegmatitic intergrowths of quartz and orthoclase are, however, very abundant.

Crystalline Limestone.—Besides the dikes mentioned, and the deposits of iron ore, the diorite includes small masses of limestone. The best locality is Lime Point, described below, but limestone is also found near the Barcelona mine on the west side of the Magdalena River and 2 miles from Lola Hill. Here, however, the field relations cannot be made out with certainty. Another small limestone mass is reported from the steep slope of the main range north of Daiquiri, and several others are found in close connection with ore in the Juragua mines.

Lime Point is situated 1,600 ft. southwest of the mine offices on the summit of the sharp ridge between Magdalena River and Daiquiri Creek. Here is a small mass of limestone almost 100 ft. in diameter and wholly inclosed in diorite, though the contacts are not well exposed. The stratification, trending north-south, is excellently shown, the beds being 1 to 2 ft. thick, with vertical dip. The rock is greenish, mottled and crystalline, with medium grain. It is irregularly replaced by yellow

garnet, a brownish green rhombic pyroxene, and some iron ore. In fact, some ore was taken from this place, but being of low grade was not shipped. A polished section showed that this iron ore consisted, like that of Lola Hill, of magnetite partly oxidized to hematite. A tunnel driven 50 ft. below the summit on the Magdalena side failed to find any continuation in depth of this limestone mass.

The significance of this occurrence in relation to the origin of the iron ores will be referred to below. It is clearly a partly contact-metamorphosed inclusion of a sedimentary series older than the intrusive diorite.

The Ore Deposit of Lola Hill

General Occurrence.—The great ore deposit of Lola Hill, from which 6,000,000 tons of high-grade iron ore have been mined, and in which much still remains, forms, broadly speaking, three lenses, partly connecting, aligned north-south, and aggregating 2,500 ft. in length. The dip is very steep to the east; the width varies up to 120 ft. Cutting squarely across the summit of Lola Hill, 600 ft. above the offices in the valley, it has been mined chiefly from a series of open cuts beginning 200 ft. above the base. Seen from a distance the hill presents a striking appearance, with its numerous benches with railroad tracks and steam shovels, and long lines of horizontal dumps where was deposited the barren rock which necessarily had to be removed from the walls of the deposits. From south to north the lenses are known as the San Antonio, nearest to the offices, the Lola, and the Magdalena. Three inclines lead down to the railroad from the open cuts and the San Antonio tunnel opens the lowest parts of the deposits from a few feet above the railroad level. This tunnel is still in magnetite ore, but the width is here only about 30 ft.

The Magdalena lens is developed by an open cut and by a tunnel approximately 200 ft. higher than the main railroad track at the offices.

Ore and Gangue.—Most of the ore consists of a dense or fine-grained mixture of magnetite and hematite, though in the lowest tunnels of the San Antonio and the Magdalena mines only magnetite occurs and this is rather coarser grained. The gangue consists of quartz, which is far more conspicuous in thin section than in the specimen. Garnet and a little amphibole, epidote, calcite, and pyrite are present in varying amounts. The ore in the Magdalena tunnel, for instance, is admixed in places with so much pyrite as to render it valueless for shipping. Copper stains are frequently seen and some parcels of oxidized copper ore have been shipped to the Cobre mine. Fresh chalcopyrite is rarely observed.

Recent analyses of iron ores from Daiquiri run as follows, in percentages:

Average of 134,000 *Tons of Ore from the Lola Mine*

Fe	59.9
P	0.019
SiO_2	9.5
S	0.33

Average of 153,000 *Tons of Ore from the Magdalena Mine*

Fe	59.2
P	0.021
SiO_2	11.3
S	0.89

Average of 42,000 *Tons of Ore from the Barcelona Mine*

Fe	58.7
P	0.03
SiO_2	11.1
S	0.4

Earlier analyses quoted by Spencer[3] show that the ore then mined averaged from 62 to 65 per cent. of iron and contained somewhat less silica. Analyses of average samples given by him run as follows:

Analyses of Average Samples of Iron Ores from the Lola Mine

	1896	1897
Fe	63.050	63.100
Mn	0.062	0.097
Cu	0.016	0.056
Ti	0.007	
S	0.048	0.072
P	0.025	0.029
Al_2O_3	0.821	0.712
CaO	0.890	1.060
MgO	0.259	0.381
SiO_2	7.585	7.225

Geological Field Relations of the Ore.—The ore, then, forms in general a tabular body in diorite, 2,500 ft. long, up to 100 or 120 ft. wide and at least 600 ft. deep. Actually it is divided in three parts by the intrusive rock and, moreover, as noted already by A. C. Spencer,[4] it is cut in many places by igneous dikes. Two of these were observed in the open cuts of the San Antonio mine. Both were decomposed, but one of them which was 6 ft. wide proved to be a granite porphyry having phenocrysts of albite-oligoclase and quartz in a microgranular groundmass with feldspar and quartz sometimes in micropegmatitic intergrowths. The rock is evidently allied to the aplites and may be considered as later than the normal diorite.

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[3] *Report on a Geological Reconnaissance of Cuba*, p. 77 (Washington, 1901).
[4] *Idem*, Fig. 12.

The exposures along the successive benches for a vertical distance of 400 ft. are excellent and show no direct evidence of surface oxidation in the way of limonite or kaolin. Each face is from 50 to 200 ft. below the original surface; the summit of the hill has been removed by the open-cut work.

In the main, the contacts between ore and the surrounding diorite are definite and the black orebodies stand out sharply against the whitish or greenish igneous rock. In detail, however, there is evidence of contact zones and intimate intermingling.

The iron ore itself is not so uniform as it appears at first glance. Generally fine grained, steel gray and giving a red streak, there are places where the admixture with quartz and pyrite becomes prominent; the quartz in irregular masses and veinlets, the pyrite in streaks of small crystals. Calcite is fairly abundant in places, but generally fills small irregular cavities or forms later crusts. More prominent than these minerals is garnet, which occurs abundantly in the ore, sometimes intimately mixed, but more often as heavy bodies or flat masses several feet in thickness. It is yellowish brown and massive, though showing in places small crystals in drusy cavities. A qualitative analysis showed it to be a normal andradite with little alumina.

The contact between ore and diorite can usually be located within 1 or 2 ft., but frequently dike-like masses of iron ore or of massive garnet 1 or 2 ft. wide project into the diorite for a distance of many feet, just as dikes of rock project into the ore.

The diorite adjoining the ore is rarely quite normal, but contains much epidote and is frequently darker and more fine grained than the typical rock, for a distance of 10 to 30 ft. from the contact. An occurrence of this kind from the lowest open cut on the San Antonio was designated in the field notes as "a heavy, fine-grained variety suggesting absorption of lime." Going west, the rock gradually becomes more normal, but the grain continues uneven and the rock contains much epidote. A specimen from this place about 10 ft. from the ore is further described below and designated as No. 7 Daiquiri.

In the deep tunnel, which extends from the railroad level into the San Antonio mine, the ore is only 30 ft. wide and consists of magnetite admixed with garnet, epidote, and vug fillings and coatings of calcite.

In the open cut of the Lola mine the exposures are similar to those of the San Antonio.

In the open cut of the Magdalena mine the ore is about 100 ft. wide and appears to be bounded by a fault on the east side. The ore is massive with but little admixture of gangue. The Magdalena mine is also opened by a tunnel about 200 ft. above the office, but on the slope toward the Magdalena River. Ore is here mined by underground methods. It is mostly coarse-grained magnetite oxidized to limonite

along fissures. No garnet was seen, but there is much pyrite in places and some of the ore is a dark-green mixture of amphibole, feldspar, quartz, and magnetite described below as No. 20 Daiquiri. A winze sunk 100 ft. below the Magdalena tunnel developed magnetite with 8 or 10 per cent. of pyrite.

Detailed Description of Ore and Gangue.—The "contact rock" (7 Daiquiri) previously mentioned was examined in some detail and proved a rather puzzling type. It is finely granular under the microscope and consists of 50 per cent. or more of amphibole and pyroxene, while the remaining half consists of a feldspathic mineral. It contains very few grains of iron ore, but a notable amount of small grains, crystals, and aggregates of titanite, which no doubt is derived from ilmenite.

There are a few equidimensional grains or rough prisms of colorless augite with normal extinction. Far more abundant is a pale-green uralitic hornblende in irregular grains or sheaves frayed at the edges, while needles of the same mineral are abundantly scattered through the feldspar, so that it would be impossible to separate the latter by heavy solutions. Apatite in small prisms is abundant. The feldspathic material consists of interlocking grains, which have low double refraction (about 0.004–5) and an index of refraction varying from 1.54 to 1.55, only one or two of these grains showing polysynthetic twinning with extinctions indicating andesine. The refraction points to oligoclase-andesine. The mineral is somewhat attacked by concentrated hydrochloric acid. A partial analysis was made of this rock by C. S. Venable and gave 51.67 per cent. SiO_2, 11.54 per cent. CaO, 2.9 per cent. Na_2O, and 0.9 per cent. K_2O. This somewhat surprising result does not agree with the normal composition of an igneous rock composed of andesine, augite and hornblende, and suggests strongly the presence of a scapolite mineral. There is certainly feldspar in the rock, but the scarcity of twinning is remarkable and the refraction is somewhat too low for andesine. On the other hand, the section shows apparently only one prominent feldspathoid mineral; and the double refraction is decidedly too low for a calcium scapolite. The analysis confirms the suspicion entertained in the field that this dark "contact zone" has absorbed lime. Very likely the amphibole contains an exceptionally large percentage of calcium, or the feldspar is, in parts of the specimen, converted to a scapolitic mineral.

A specimen collected from the other, or eastern, contact shows a decided similarity to the one just described, although the contact rock on this side is not so well developed as on the other side. It is a greenish, fine-grained rock. In this section it shows an interlocking feldspar aggregate with some little quartz. Only one or two of the feldspar grains are twinned and give angles of extinction indicating andesine; the grains are in great part cloudy and filled with chlorite foils and horn-

blende needles. Fibrous hornblende is abundant in frayed prisms. There is no black iron ore, but some secondary titanite.

Much of the material mined is massive iron ore, and sections of this show little except a few included small grains of quartz, garnet, or pyrite. The most common gangue mineral is quartz, and this occurs intimately mixed with the iron ore. Analyses quoted on p. 46 indicate an average of 10 or 11 per cent. of silica in the ore now being mined, and most of this silica is present as quartz. A section of typical quartzose ore is shown in Fig. 1. The specimen (18 Daiquiri) comes from the Magdalena open cut. The iron ore, which is magnetite with hematite rims,

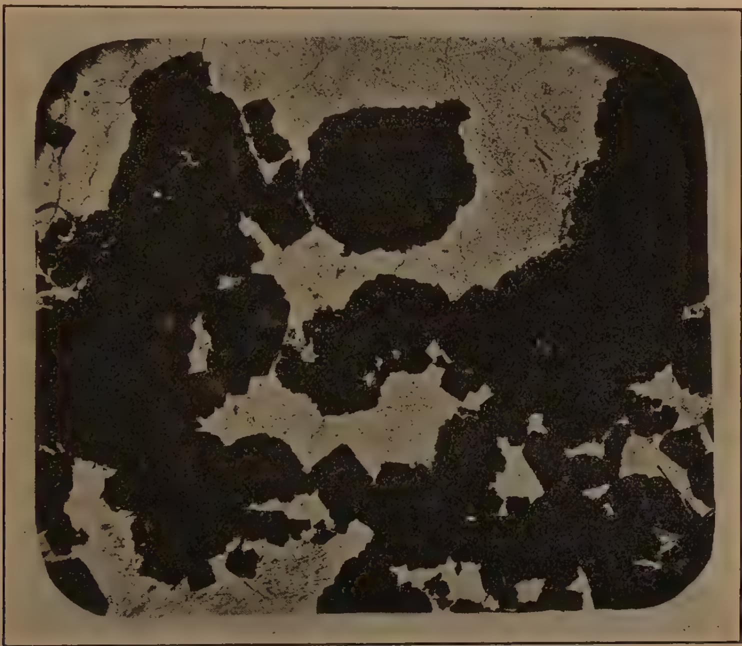

Fig. 1.—Photomicrograph of Siliceous Ore from Magdalena Open Cut. Black, Magnetite with some Hematite; gray, Quartz. Magnified 30 diameters.

forms irregular or rounded masses often bordered by projecting crystal faces, as may be seen in the figure. The quartz, which is coarsely granular, fills the interstices and contains a few small prisms of apatite. In places the impression is that of magnetite shattered and filled with quartz veins. There is little doubt that the magnetite is the earlier mineral. The section contains a few cubes of pyrite, which is distinctly associated with the quartz, and is later than the magnetite.

Another type of quartz magnetite rock is shown in Fig. 2; the specimen (21 Daiquiri) was taken from the deep San Antonio tunnel. The magnetite here forms small crystals and striking dendritic groups, all

imbedded in granular quartz, which also contains a considerable number of minute prisms of apatite. From this section the distinct impression is gained that the magnetite crystallized in a yielding medium, which later consolidated to granular quartz.

The presence of andradite garnet has been mentioned, but this mineral is less commonly seen in the shipped ore, for it forms large "bars" and masses which are easily sorted out. The ore shipped now contains only about 1 per cent. CaO, but the body as a whole certainly contains 3 or 4 per cent. of this constituent, present as calcite and andradite. Sections of iron ore often show small inclosed grains of garnet

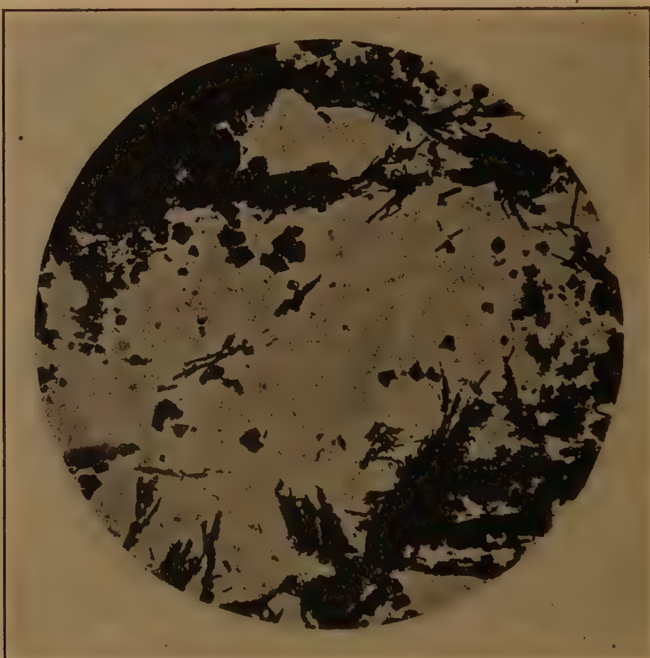

Fig. 2.—Photomicrograph of Siliceous Ore from the San Antonio Tunnel. Black, Magnetite; white, Quartz. Magnified 30 diameters.

and calcite, and both of these sometimes contain brownish green prisms of amphibole. The garnet rarely shows crystal outlines or the zonar birefringency so common in contact-metamorphic deposits. The calcite occurs either as grains in which, as stated, prisms of amphibole have developed, or as filling of small cavities, in which case it shows concentric deposition lines.

A type of lean ore which is rare at Daiquiri, but not uncommon at Juragua, is shown in Fig. 3. The specimen (24 Daiquiri) was taken from the Magdalena tunnel at the north end of the ore zone and occurs in heavy magnetite. The rock is coarse grained and dark green, and

under the microscope proves to be a granular mixture of monoclinic pyroxene, magnetite, quartz, and feldspar. The magnetite forms small crystals and rectangular bars, which by their arrangement indicate a coarse dendritic texture. The magnetite seems to be the earliest mineral. There is much granular or roughly prismatic augite, also some quartz, and a few grains of andesine feldspar, one of which is shown in the figure. While it is possible that this is an igneous rock, its interpretation as a replacement product of limestone would seem more probable.

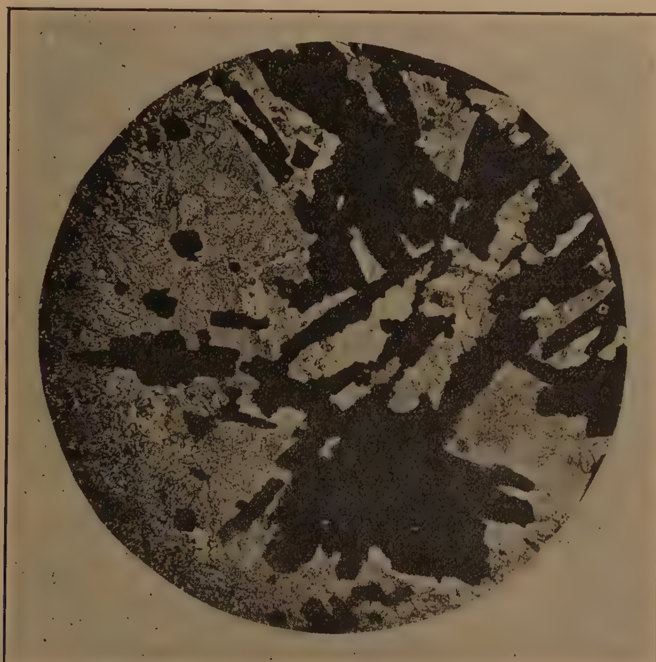

FIG. 3.—PHOTOMICROGRAPH OF LEAN ORE FROM MAGDALENA TUNNEL. Black, Magnetite; gray with granular surface, Augite; light gray, Quartz, with one grain of Andesine in center. Magnified 30 diameters.

Metallographic Examination.—Polished faces of the ore are difficult to prepare by reason of pitting and failure to take perfect polish. The sections studied all showed magnetite and hematite and the same relationship between the two minerals. Etching with hydrochloric acid accentuates the difference between magnetite and hematite a little better, both being slightly attacked, the latter more than the former. The magnetite, which predominates in most cases, has a dark-gray color with a tinge of red. The etching often brings out a system of lines, or two systems crossing at, say, 60°. These lines are caused by the octahedral parting in magnetite. The hematite is brighter and light gray in color and the polished faces are full of small pits.

The hematite occurs as a border of varying width around the magnetite and the contact between the two is usually wavy and irregular. Tongues of hematite, sometimes highly irregular in outline, enter along all possible openings, such as cracks and contacts between grains. The hematite has also a strong tendency to penetrate the magnetite along its parting planes, and in several places the parting is clearly brought out by the bright thin lines of hematite outlining a series of rhombs. In other sections the magnetite grains are penetrated by an exceedingly delicate, irregular network of hematite visible only with the highest magnification.

The pits in the hematite are probably caused by the local development of a softer or earthy facies of the mineral.

All gradations are present in the ores. Some are mostly magnetite, with very little hematite covering the grains as a thin film or penetrating them as a delicate filigree work. Others consist of about equal amounts of both minerals, and finally some appear as scaly hematite which are not magnetic and contain only a few residual grains of magnetite.

Very little limonite is present and, when found, bears no relation to the amount of hematite. In fact, the hematite appears to become hydrated only with great difficulty and most of the limonite seen, which came from the Magdalena tunnel, is a direct product of oxidation of magnetite.

The metallographic microscope shows that the pyrite is distinctly later than the magnetite; it usually forms cubes from 1 to 8 mm. in diameter. In many sections, which show mostly hematite, pyrite crystals remain fresh without any oxidation.

Genesis of the Deposits

From the above it is clear that the primary iron oxide of the deposits at Daiquiri is a magnetite, which subsequently has been altered more or less completely to a hematite. The discussion of genesis may therefore be divided into two parts: The first relating to the origin of the magnetite; the second to its subsequent hematitization.

The mineral association of the magnetite, particularly the presence of much garnet, shows that it originated under high-temperature conditions.

On the other hand, the hematitization is probably a low-temperature process developing gradually under the influence of oxidizing atmospheric waters.

The principal previous contribution to the question of the origin of the Daiquiri ores comes from the pen of A. C. Spencer of the U. S. Geological Survey.[5] He considers three views:

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[5] *Report on a Geological Reconnaissance of Cuba*, pp. 79 to 82 (Washington, 1901).

1. The orebodies are an igneous product of processes of differentiation in a molten diorite magma.

2. The orebodies are a product of contact metamorphism exerted by the diorite on included masses of limestone.

3. The orebodies form a part of an older limestone-schist series, which was disrupted by the intrusion of the diorite. In this case the orebodies were formed by metamorphic processes in these sedimentary rocks long before the intrusion of the diorite.

Spencer believes the first and second theories are not very plausible, but is inclined to accept the third. He says:[6]

> "It is necessary to suppose that the orebodies of the metamorphic series were originally widely distributed and of immense size. . . . A mass of many million tons weight floated upward by the buoyant effect of molten rock in motion from the interior towards the surface of the earth, is the only conception which adequately accounts for the mode of occurrence of the orebodies of the Magdalena and Lola mines at Daiquiri; while, though less strikingly shown, at Firmeza (Juragua) it is likely that the masses of schist, marble and ore have been likewise actually suspended in the molten lava."

Regarding first the theory of origin by differentiation in the diorite magma, we fully agree with Spencer that there are weighty objections against it. The association of ore with andradite, the lack of transitions between ore and diorite, the very small quantity of phosphorus present, and, last but not least, the definite character of the orebodies of *inclusions* —all these considerations militate strongly against this theory.

The third theory of Spencer presupposes a series of limestone and hornblende and epidote schists, which contained immense masses of magnetite, probably formed by regional metamorphic processes, and of which the present masses are but small remnants. The assumption of such a series of metamorphic rocks does not seem fully proved. At Daiquiri there is bedded limestone, but no schist whatever, and from Spencer's description of Juragua it does not clearly appear that the "schist" at that place really constitutes a conformable series with the limestone. The formation of such tremendous masses of magnetite, as must be assumed to have existed, by regional metamorphic processes is wholly improbable; we believe no such deposits are known. If they were formed by igneous metamorphism there seems to be no evidence of the old pre-dioritic intrusive which caused the orebodies to form. For these reasons it seems necessary to reject the theory of an older iron-bearing series of sediments.

There remains the second theory, accounting for the large masses of magnetite included in the diorite by contact-metamorphic action of the magma on an older limestone, by which the latter has become almost entirely replaced by magnetite derived from magmatic emana-

[6] *Idem*, p. 82.

tions rich in iron. While it is freely admitted that the genesis of the Daiquiri deposits is difficult to explain, it will be shown that the view outlined in the previous sentence has much in its favor.

In the first place, the relations at Lime Point prove that a bedded limestone formation existed here before the intrusion of the diorite and that the fragment preserved at this place has been partly metamorphosed by the diorite, with the development of garnet, pyroxene, and magnetite, the latter in not inconsiderable quantities. The composition of the main orebody is that of a contact-metamorphic deposit with much andradite, garnet, quartz, and magnetite and some residuary calcite. It has been shown that grains of residuary calcite inclosed in magnetite contain prisms of amphibole, indicating exposure of these grains to metamorphosing influences. The only puzzling question is, why larger masses of calcite have not escaped the metamorphism, for in most deposits of this kind the change to ore is rarely complete. It may also seem strange that a smaller mass like that of Lime Point should have been partly metamorphosed, while the much larger mass of Lola Hill suffered almost complete replacement, but the capricious character of such metamorphism is well known.

The chemical composition of the ore is exactly that which would be expected in a contact-metamorphic deposit. The percentage of phosphorus is very low, while there is much pyrite in places and some chalcopyrite. The little phosphorus present is contained in the apatite prisms, which are inclosed in the quartz, and this apatite is believed to indicate that the quartz is not an infiltration which took place at low temperature, but that it was of magmatic origin and deposited from very hot solutions. Still another argument in favor of a genesis by replacement of limestone is furnished by the abnormal percentage of lime in the dioritic contact rock adjoining the deposit in many places. This percentage is much too high for a normal quartz diorite, or in fact for any igneous rock of the general type indicated by the thin sections, and points without doubt to a migration of lime outward into the igneous rock.

The enormous amount of calcium and carbon dioxide which must have been carried away and the correspondingly large additions of iron oxide, silica, and sulphur almost strain the imagination, it must be confessed, but students of contact metamorphism are well acquainted with similar and well-authenticated examples elsewhere. That there must have been a most energetic interchange of constituents between the limestone and the igneous rock is indicated by phenomena such as dike-like masses of andradite and magnetite along the contacts of the orebody. The dikes of igneous rock intruded into the iron ore appear to be somewhat different from the diorite and are either granite porphyry or aplite; that is, complementary dikes of a later generation.

The violent interchange of constituents along the contacts suggests that the diorite was fluid when the metamorphism took place.

Regarding the oxidation of the magnetite to hematite there is but little to add. Hematite appears to be more abundant close to the surface than in depth, hence an increased probability that descending surface waters effected this work. The change from magnetite to hematite involves but a slight addition of oxygen. It is certain that the amount of this gas was not large enough to oxidize appreciably the pyrite at a distance of about 100 ft. below the surface. The occurrence of fresh pyrite in almost completely hematitized magnetite is an interesting feature that deserves further investigation.

In explaining this mode of oxidation, the rapid erosion of this region should be borne in mind. At the surface, magnetite probably tends to become converted to limonite. It is likely that the conversion to hematite takes place at greater depth, below the water level, where the amount of oxygen available for the process is much smaller. According to this view the hematitization of this deposit took place at a depth of several hundred feet below the original surface, and as the conversion of hematite to limonite is exceedingly slow, the hematite acted as a protecting cover for the magnetite when erosion had lowered the surface and had exposed the hematitized portion of the deposit to the action of surface waters richer in oxygen.

The conditions under which magnetite alters to hematite are imperfectly known. It is suggested that a warm climate is favorable to this change.

The Ores of the Juragua Mines

General Features.—Unfortunately, time did not permit a visit to the Juragua mines, situated about 8 miles west of Daiquiri and connected by a railroad with the shipping point in Santiago Bay. From the available descriptions by Kimball and Spencer the deposits are on the whole very similar to those at Daiquiri, and mainly form irregular masses inclosed in an igneous intrusive rock. Spencer mentions a schistose rock closely associated with the iron ore at Firmeza mine and also describes crystalline limestone in close contact with the ore. The magnetite, according to a figure in his paper, apparently cuts across the limestone and both are intersected by narrow porphyry dikes. The present manager of the Juragua mines, De B. Whitaker, stated, in a personal interview, that limestone is found in at least two of the Juragua mines and the garnet also occurs in the ore. The deeper levels are said to yield chiefly magnetite.

A series of specimens was collected from the large stock pile at the shore of Santiago Bay and was examined with some care. No garnet or limestone was found in the stock pile. The ore is in large part the

same magnetite, partly converted into hematite, which characterizes the Daiquiri deposits, but there was also more or less granular magnetite closely associated with dark-green fine-grained igneous rocks. Many specimens consisted of a mixture of magnetite and actinolite or of magnetite and pyroxene. Epidote is abundant, mostly as veinlets distinctly later than the ore; intersecting the ore are also many veinlets of quartz and small dikelets of a quartzose granular rock.

The Juragua mines, which have been worked since 1884, have yielded about 6,000,000 tons of iron ore.

Detailed Descriptions.—The igneous rock mentioned above is fine grained and dark green. The thin section shows it to consist mainly

Fig. 4.—Photomicrograph of Lean Ore, Juragua Mines.
Black, Magnetite; gray with granular surface, Augite. Magnified 70 diameters.

of a basic feldspar, mostly labradorite in larger and smaller prisms and shredded prisms and needles of green hornblende. The rock is probably an altered diabase porphyry. Accessories are magnetite in small grains, apatite, and quartz. Calcite and epidote form the secondary products.

Other dark-green rocks differ radically from the specimen just described and recall certain products of contact metamorphism. The principal constituent in these is augite in grains (Fig. 4), often small and rounded or irregular, more rarely, roughly prismatic. The other constituent is magnetite in irregular grains, apparently earlier in origin than the

augite, which penetrates it in veinlets. In one section a single grain of a basic feldspar, probably bytownite, was found, while in another section feldspar was a little more abundant and none of the lamellæ showed extinction of more than 21°, which would indicate andesine-labradorite. Epidote and quartz are secondary and only sparingly present.

Other specimens of dark-green lean ore contain much actinolite in bands or bladed aggregates, and more or less quartz in granular masses. Calcite is frequently present in small grains and veinlets or more rarely

FIG. 5.—PHOTOGRAPH OF BANDED MAGNETITE ORE, JURAGUA MINES. Black, Magnetite; light gray, Actinolite with some Quartz. Natural size. Specimen polished on two sides.

in larger cleavable masses about ½ in. in diameter. Epidote is fairly abundant in larger masses, grains, and veinlets.

A number of ore specimens were cut by sharply defined veinlets of quartz with more or less perfect comb structure and frequently a central band of epidote or calcite; in one case the quartz veinlet contained well-formed crystals of magnetite and pyrite. Pyrite forms small granular streaks in some iron ore and occasionally a little chalcopyrite with malachite is encountered.

One specimen of iron ore contained several dikelets apparently of

igneous origin and from 2 to 10 mm. wide. They consist of a granular mass of pink feldspar, quartz, and scattered grains of magnetite.

A peculiar banded ore may deserve special mention. Its appearance is shown in Fig. 5, which is a photograph in natural size of a specimen polished on two sides. The dark bands are magnetite, the lighter bands mainly actinolite, the two separated by curving but sharp boundaries. The needles of actinolite are mostly perpendicular to the banding. In some of the wider bands the actinolite is partly converted into asbestos. A thin section shows that much quartz in fair-sized, interlocking grains is associated with the actinolite and that some quartz is also present in the magnetite bands. The magnetite forms small irregular grains and aggregates; in places, however, rectangular outlines are seen, suggesting crystal form. The texture of this specimen strongly suggests diffusion banding, such as might form in a material, a limestone, perhaps, freely penetrated by hot iron-bearing solution. Somewhat similar textures have, we believe, been described from the limestone and "skarn" ores of central Sweden. Some of the iron ore from Juragua is distinctly reddish and scaly like normal hematite or seamed by red hematite. Most of it is, however, dark gray and fine grained and shows a red streak when scratched. The coarse granular magnetite is less common.

Under the microscope quartz is seen to be a common constituent. Some of the iron ore forms rounded grains which are intimately intergrown with the interlocking quartz. Much of it, however, is in the form of elongated rectangles and needles crossing each other and the quartz grains in all directions or gathered into radiating clusters or sheaves.

The metallographic microscope shows that the Juragua ores, like those of Daiquiri, are intricate mixtures of primary magnetite with secondary hematite.

The normal ore appears as a fine-grained black material with a few bright surfaces of micaceous hematite, and streaks and irregular patches of granular pyrite, some of which show striated crystal faces. The ore is magnetic, although not as strongly so as pure magnetite, and gives a red streak. The study of the polished sections shows that the greater part of the ore is granular magnetite. Hematite occurs as a rim around the magnetite grains, and penetrates them in an exceedingly intricate manner. Sometimes, where no visible opening exists, it penetrates as vein-like threads forming a beautiful, lace-like pattern.

Probable Genesis.—The iron ores of Juragua are then partly hematitized magnetites with gangue of quartz, augite, actinolite, and epidote. Igneous rocks of the diabase porphyry type occur in the district. The relationship of the magnetite to the augite and actinolite strongly suggests a contact-metamorphic origin of the ore and this conclusion is strengthened by the presence of chalcopyrite in the ore.

Literature of the Daiquiri Mines

James P. Kimball: The Iron Ore Range of the Santiago District of Cuba, *Trans.*, xiii, pp. 613 to 634 (1884–85).

———: Geological Relations and Genesis of the Specular Iron Ores of Santiago de Cuba, *American Journal of Science*, Series 3, vol. xxviii, pp. 416 to 429 (1884).

F. F. Chisholm: Iron-ore Beds in the Province of Santiago, Cuba, *Proceedings of the Colorado Scientific Society*, vol. iii, pp. 259 to 263 (1888–90).

C. Willard Hayes, T. Wayland Vaughan, and Arthur C. Spencer: *Report on a Geological Reconnaissance of Cuba*, made under the direction of General Leonard Wood, Military Governor, pp. 69 to 83 (Washington, 1901).

Arthur C. Spencer: The Iron Ores of Santiago, Cuba, *Engineering and Mining Journal*, vol. lxxii, pp. 633 to 634 (Nov. 16, 1901).

———: The Mayari and Daiquiri Iron Ore Mines. Articles published in *The Iron Age*, vol. lxxx, pp. 421 to 426 (Aug. 15, 1907) and vol. lxxxi, pp. 1149 to 1157 (April 9, 1908).

Discussion

Waldemar Lindgren, Boston, Mass.—I happened to go down to Cuba rather unexpectedly in the winter of 1914, and spent a few very interesting days at Daiquiri and Mayari. To C. F. Rand and his associates I am greatly indebted for permission to visit the mines and for many courtesies received. A few months after that, I met Professor Kemp who informed me that he had just returned from a geological study of Daiquiri. We talked over what to do in a case like that. We both liked to publish our opinions, and so finally agreed to write two papers entirely separately, I not knowing any of Professor Kemp's conclusions,[1] and *vice versa*.

We were rather interested in seeing what would happen. Of course, as might be expected, we disagreed on some points; not, of course, that we ordinarily disagree, but simply because when two men work in the same field and with different ideas, probably they are apt to vary slightly in their conclusions.

Some years ago A. C. Spencer went down there for the military interim government and published the results of his study. His view of the origin of the deposit differed from that of Professor Kemp and from mine. So here are three different results obtained by three different geologists. Such differences in opinion on obscure subjects sometimes tend to bring geology into disrepute. It is perhaps not surprising that different opinions have been expressed, for this occurrence of iron ore is one of the most difficult to explain that I have ever seen.

Max Roesler, Great Neck, L. I.—I had the opportunity this summer of spending about two months down at the Juragua deposits, and studied

[1] See The Geology of the Iron-Ore Deposits In and Near Daiquiri, Cuba, this volume *Trans.*, pp. 3 to 39.

them with the idea of writing a paper on them, which I hope will come out later.

The interpretation of the occurrence at Ocania Mine is the thing which particularly struck me, in the last paper.

At the Ocania Mine we may have these large blocks of limestone which have been utterly and entirely altered to a remarkable extent.

The limestone has been engulfed in a coarse porphyry which has phenocrysts up to as much as an inch in diameter. I think the engulfing of the limestone happened long before the period of mineralization. The mineralization at the Ocania Mine has affected the limestone somewhat, has undoubtedly formed a certain amount of ore in the limestone, but the largest part of the ore at the Ocania Mine can be traced into this coarse porphyry; you can get all the stages from coarse porphyry to an almost pure magnetite, and I believe and hope to show that to a large extent this ore is a replacement of this coarse porphyry. In the same way in the rest of the mines at Firmeza, the greater part of the ore lies in igneous rocks, some of them coarse-grained, rather more basic than the ordinary diorite, I think, and some of them distinctly fine-grained and fragmental, and the ores almost always are associated with aplitic, highly acid dikes.

These dikes, even when as small as 2 in., in cutting through these igneous rocks have alongside of them a rim of magnetite and lime-iron garnet.

The mineralization can be traced right into the unaltered diorite. The Daiquiri deposits I visited only casually, but in the Firmeza districts the deposits were localized in the igneous rocks, in places where they had absorbed a certain amount of lime from neighboring limestone.

Benjamin B. Lawrence, New York, N. Y.—I have observed with a great deal of interest these deposits of iron which have been discussed, having been engaged in mining in Cuba for the past seven years. In prospecting in Cuba, we have run across a number of very interesting occurrences of iron, one especially, in Santa Clara County, about 30 miles from Cienfuegos, where we found a contact deposit of iron in an altered schist, and limestone; of an average width of about 40 ft. The gossan outcrop breaks off in large boulders, and showed the presence of copper. We diamond-drilled this deposit to the depth of 185 ft. At this depth we penetrated the gossan and encountered iron pyrites containing 50 per cent. sulphur.

After failing to find the recurrence of what we thought would be a duplicate of what happened at Ducktown (a secondary copper enrichment of chalcocite, between the hematite and the pyrite), and as the pyrite was running about 1 per cent. of copper, after perfecting our titles to the property, we temporarily abandoned it.

It is an ill will, as you know, that brings no one good. The result of

the war has been to stimulate the production of pyrites for sulphuric-acid manufacture so that this pyrite deposit has now passed into hands of the Davison Chemical Co. of Baltimore, who intend to develop it for the pyrites. There are other interesting deposits of iron in Pinar del Rio, in the extreme western end of the island, many of them underlain by pyrites. These are more irregular in character than the one I have described in Santa Clara.

In Santiago we are developing the old "Cobre" copper mine, and have reached a depth of 1,300 ft. There we followed down on two well-defined veins in the diorite, and at a depth of 600 ft. the quartz gangue in one portion of the vein was entirely replaced by gypsum.

We found no evidence of limestone *in situ* in our working of this deposit, yet on the 600-ft. level in one stope, which ran for a distance of 200 ft., the quartz gangue was substituted by gypsum.

All this may have some bearing upon the deposits under discussion. The island of Cuba, I might say, is just beginning to feel the influence of American engineers who have gone down there, and I think there is much which may come of these investigations.

Louis C. Graton, Cambridge, Mass.—One of the subjects in this paper which particularly interests me is the finding of an apparent conversion of magnetite into hematite, and this below the water level and in the presence of sulphides.

Everyone, I suppose, has seen indications as to how stable a mineral magnetite is and how it refuses to undergo oxidation even though a part of its iron is in the ferrous state, and though an addition of so small an amount of oxygen as 2.5 per cent. would suffice to convert it entirely into the ferric state, *i.e.*, hematite. Black sands consisting largely of magnetite, like those on some of our Pacific beaches, represent the heavy, chemically resistant particles concentrated from great volumes of rock which have been worn down by the slow process of erosion. These sands, agitated by the waves year after year, no doubt century after century, and alternately wet and dry, would seem to be under conditions especially favorable to oxidation. Yet neither in the long process of accumulation, nor the long period during which it is buffeted about on the beaches does the magnetite appear to undergo appreciable oxidation. Sandstones at least as old as Paleozoic contain magnetite which gives every indication of having been an original constituent, yet is fresh and unoxidized at the present day. Numerous indications such as these lead to the belief that magnetite is especially difficult of oxidation.

On the other hand, the readiness with which the sulphide minerals (and related arsenides, sulpho-salts, etc.) oxidize needs no emphasis. Though pyrite is not one of the most rapidly oxidized of the sulphides, we know that rocks containing but a small amount of that mineral are avoided as building stones because of its rapid oxidation, with resulting

weakening or at least discoloration of the stone. And except in locations of exceedingly vigorous present erosion or recent glaciation we never find sulphides near the surface without plentiful evidence of their oxidation. Indeed, wherever we do find unoxidized sulphides, we regard their presence as the most delicate and conclusive proof of the absence of direct oxidation at that place.

Evidences of the difficulty with which oxidation[2] proceeds below the water level are common. Reference need only be made to the prevalence of "free-milling" or oxidized ores, of gossan, and of leached ground above that horizon, and of "rebellious" or sulphide ores below. Though there are exceptions to this general rule, they are indeed exceptional, and are in part explainable in other ways than that oxidation has actually taken place below the water level. Moreover, in a number of the places where it may be assumed that the process of direct oxidation has indeed been working under water, it is found that oxidation is less vigorous and less complete than it is in the same deposit above the water level. Finally, direct oxidation below water level is generally able at least to attack only the most easily oxidizable materials, namely, the sulphides; minerals of more stable character largely or wholly escape its influence.

Since the paper by Lindgren and Ross was published, material from another region, studied by W. L. Whitehead, shows the association of magnetite, hematite, and sulphides, that corresponds to Professor Lindgren's description, and Mr. Whitehead and I are satisfied as to the correctness of his explanation that the hematite has been derived by alteration from the magnetite. But I do not feel convinced that such alteration is the result of oxidation, at least of oxidation in the usual sense, *i.e.*, by oxygen derived from the atmosphere.

If it were only a question of magnetite oxidation, one might believe that oxidizing conditions had in this particular instance been so extraordinarily intense as to accomplish this difficult result. If it were a question of magnetite oxidation far below water level, I should feel that all precedents and indications must be upset to permit it; though the presence of oxidized magnetite below water level might be accounted for by the assumption that the alteration had taken place under especially vigorous oxidizing conditions above the water level which later had risen and immersed the oxidized products. But when it comes to a question of magnetite oxidation, not only far below the water level but also in contact with sulphides that give no hint of even the slightest oxidation themselves, I find myself unable to imagine conditions under

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[2] I refer here to direct oxidation such as yields predominantly oxygen-bearing compounds, like limonite, malachite, anglesite, etc.; not to indirect oxidation such as that which converts pyrite to chalcocite, with accompanying liberation of $FeSO_4$ and H_2SO_4.

which such a peculiar and apparently anomalous result could have been accomplished.

More reasonable would seem to me the view that the partial conversion of magnetite into hematite has nothing to do with surface alteration but is a deep-seated change marking a readjustment of equilibrium that took place within the period of primary ore formation.

Harrison Souder, Cornwall, Pa.—I have listened to this discussion with a great deal of interest, because I happen to have the pleasure of operating a magnetic iron ore mine with characteristics somewhat like those of the Cuban deposits.

I refer to the Cornwall Ore Banks at Cornwall, Pa. It occurs to me that a study of the Cornwall deposit might throw some light on the Cuban deposits. At Cornwall we have a foot wall of diabase, a hanging wall of limestone, with the ore between. This ore usually lies in definite bedded planes. The hanging wall is a metamorphosed limestone, no fossils have been discovered in it that I know of. From it fingers of limestone and shale project into the ore. You can see where the limestone changes into ore and back again to limestone. There are also cross dikes of diabase cutting through the ore from the foot wall to the hanging and on into the hanging wall of limestone. There were two of these dikes but one has been dug away. The existing one is now just about 2 ft. thick through the ore. Entering the hanging wall, it divides into several stringers about 2 in. thick. Native copper has been found in contact with these dikes and the limestone.

We also find a small amount of hematite mostly at or near the contact with the limestone.

Waldemar Lindgren.—In regard to the magnetite altering into hematite, I would like to say a few words. I recognized fully that it is a very unusual thing, but after all, haven't we got martite? What is martite?

In the pseudomorphs called martite the magnetite has been changed into the hematite, so why should we be so surprised at finding magnetite going over into hematite in a large deposit?

As far as the oxidation below the water level is concerned, I admit that that too is peculiar. I certainly was surprised to find those processes going on without the pyrite being affected, but one thing that guided me in my conclusion, and I do not say that conclusion is correct (it is merely tentative), was the relation of the magnetite and the hematite to the depth below the surface. Near the top of the deposit there was no question but that there was more hematite than in depth. The lowest levels of the mine showed almost entirely magnetite, while some of the stuff from near the surface is practically all hematite. That is pretty strong evidence, and as far as alteration descending below

the water level is concerned, we had ample evidence of that. There are many places where oxidation proceeds to a considerable distance, 200 or 300 ft. below the water level.

The question has another bearing; that is on the analysis of the magnetite. Some of my friends down at the Geophysical Laboratory in Washington have been analyzing this magnetite from Daiquiri, and they came to the conclusion that there was a solid solution, namely, a solid solution of magnetite in ferric oxide—they rather run to solid solutions down there. In this present case the material they analyzed was magnetite with extremely finely divided secondary hematite in it, so that, of course, they could not get the ordinary and normal percentages corresponding to magnetite.

Charles P. Berkey, New York, N. Y.—I do not yet see any reason why both of them cannot be primary. I do not know of any good reason why hematite and magnetite could not go together. Intergrowth is the word Mr. Lindgren uses, and intergrowth is rather characteristic of contemporaneous development.

Waldemar Lindgren.—If I said that, I did not convey my exact meaning; it is not an intergrowth, it is a replacement. Hematite starts out from the boundaries of the grains, from the accidental cracks and from the planes of separation of cleavage in the magnetite, so that these planes were there when the alteration started. That is not the way alteration under the influence of ascending solutions usually works, but it is the way alteration in descending solutions does work. It replaces and works into the magnetite and forms the most delicate filigree work, but always connected with those small cracks or openings from which the solutions work.

Alfred C. Lane, Tufts College, Mass.—Why might there not have been some replacement of the limestone by soft iron ores, before the contact metamorphism?

Waldemar Lindgren.—That is in fact the theory of Mr. Spencer. He assumed that the iron ores were a replacement of limestone, and that they formed long before the igneous rocks broke up the limestone and engulfed that, as well as the inclosing masses.

Alfred C. Lane.—That is very nearly it; but when I teach my students, I tell them that an ore deposit is generally a complex geological phenomenon. The question is whether or not there may or may not have been more or less replacement of the limestone by iron ore before the contact metamorphism, which is perhaps the most important action. Was there any possibility of a preliminary partial concentration of iron ore?

Waldemar Lindgren.—Do you mean before the igneous intrusion?

Alfred C. Lane.—Yes.

Waldemar Lindgren.—Yes. That might be possible, of course; but I can say it would be argued that it would be unusual to find any such large deposit of iron ore which existed previously and which by accident had been singled out by the intrusion. It is possible, of course, but it seems to me the burden of proof would be on those who assume such theories.

John D. Irving, New Haven, Conn.—This discussion dealing with the growth of magnetite and hematite is rather interesting to me, because at Leadville, Col., some peculiar and rather large deposits of magnetite mingled with sulphides, unquestionably of contact-metamorphic origin, have been found. In the lower levels where those ores are found, this supposedly solid magnetite, which is exceedingly dense and contains mingled sulphides, shows no traces of oxidation whatsoever, although far below the reach of atmospheric alteration. The ore is made up of a very fine-grained intimate intergrowth of magnetite and hematite. This would seem to me to indicate, perhaps, that the surface action is not essential in a contact-metamorphic deposit to the development of this peculiar growth of magnetite and hematite. Possibly this may also be true of the occurrences in Cuba, although I have not been there myself. I remember reading somewhere that the only condition under which a man is really qualified to speak about, or discuss a deposit, is that he should not have seen it.

Louis C. Graton.—Reference by Professor Irving to the presence in Leadville of this type of alteration of magnetite to hematite in association with pyrite below any indications of surface oxidation leads me to add that the material of this sort which I recently mentioned came from limestone deposits in the Clifton-Morenci district. May it not be significant that in the Cuba, Colorado, and Arizona occurrences, the deposits are all of contact-metamorphic origin? For in contact-metamorphic deposits both magnetite and hematite are common, and not infrequently, as shown by its appearance as seams along cracks in both the invaded and the invading rock, the specularite is found to be one of the latest products of the process, distinctly of later crystallization than the magnetite. Under such circumstances would it be unreasonable to conceive that the earlier formed magnetite might partially break down to the material closely similar in composition, but more stable in the late stages, namely, specularite?

Such an explanation, it seems to me, would account for most of the features observed in this magnetite-hematite-sulphide association. Adjustment of equilibrium such as this might take place in varying degree in different parts of the orebody; it might conceivably increase with decreasing intensity of temperature or pressure, thus possibly account-

ing for the observation at Daiquiri that hematite is more plentiful in the vicinity of the present surface and decreases downward. Or this relation might be due to the superimposition of ordinary, near-surface oxidation upon the deep-seated type of alteration here suggested.

My chief interest in the subject is to learn, if possible, whether this alteration of magnetite to hematite in the presence of sulphides is really due to oxidation, and if so, by what conditions it was brought about. My remarks are intended simply to indicate the desirability of further enlightenment on these important questions.

Waldemar Lindgren.—There is very little difference between hematite and magnetite as to the amount of oxygen in each mineral.

Max Roesler.—It seems to me rather difficult to believe that this theory of the surface action is based on a mere coincidence. The fact that there is this definite arrangement of the increase of hematite above and increase in magnetite in depth is too consistent and widespread to be fortuitous when you consider that the Daiquiri mines at which Dr. Lindgren observed it, and the Juragua and the Ocania Mines, stretch over a distance of about 15 miles, and in each case the same arrangement seems to hold good.

The Genesis and Relations of the Daiquiri and Firmeza Iron-Ore Deposits, Cuba

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I. THE GENESIS OF THE DEPOSITS

THE ore deposits at Firmeza have been worked continuously since 1884; those at Daiquiri since 1895. It is surprising, therefore, that they have not been the object of careful geologic study until quite recently. It is true that considerable attention was paid to these deposits by A. C. Spencer, in 1901, in the report on the geology of Cuba, by Hayes, Vaughan, and Spencer,[1] but this was only incidental to a hasty general reconnaissance of the Island. As far as the Daiquiri deposits are concerned, this gap has been filled by the appearance of the two papers by Prof. James F. Kemp[2] and by Prof. Waldemar Lindgren and Clyde P. Ross respectively.[3] Max Roesler, first holder of the Emmons fellowship, made a detailed study of the Firmeza deposits during the past summer, so that soon we can expect a full description of them. Consequently, there is no demand at the present time for a description of the deposits from other sources, and that is not the purpose of this paper. The papers on Daiquiri, however, leave unsolved problems in regard to the age of the rocks of the district and certain phases of the genesis of the deposits, and it is with the hope of aiding in the solution of some of these problems that this contribution is offered.

Kemp, and Lindgren and Ross agree in assigning these deposits to the type of contact metamorphic deposits. They disagree in part on the question of their localization. The deposits occur in an extensive area of diorite which includes numerous blocks of limestone, and sometimes are found in the contact zones between these blocks and the diorite, and in other instances are completely enveloped in the diorite, showing no apparent connection with the limestone. The latter class includes

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[1] C. Willard Hayes, T. Wayland Vaughan, and Arthur C. Spencer: Report on a Geological Reconnaissance of Cuba (1901).

[2] James F. Kemp: The Geology of the Iron-ore Deposits In and Near Daiquiri, Cuba. *Trans.*, This volume, pp. 3 to 38.

[3] Waldemar Lindgren and Clyde P. Ross: The Iron Deposits of Daiquiri, Cuba. *Idem*, pp. 40 to 59.

the larger and more important orebodies. With regard to the genesis of the former, there is no question; the difference of opinion hinges on the explanation of the latter. Kemp believes that they represent not replacements of limestone, but alterations of the diorite along lines of crushing and faulting by magmatic emissions from some underlying cooling igneous mass; whereas, Lindgren and Ross look upon them as extreme cases of the first class in which the limestone has completely succumbed to the metamorphosing influences.

Probably the first to recognize the nature of contact metamorphic deposits was B. von Cotta, in 1864, in his work on "Die Erzlagerstätten im Banat und Serbien." Deposits of the same nature were subsequently described in the Christiania region; in 1879 they were established as a special class by A. von Groddeck in his "Die Lehre von den Lagerstätten der Erze," under the name "Typus Christiania," by which they were known for a long time. But even as late as 1900, the number of examples referred to this type was not large, and, for the most part, the deposits were of little economic importance. Indeed, at that time so little importance was attached to the group that Spencer, although he considered contact metamorphism as a possible mode of origin of these deposits, readily dismissed it as not very plausible. The real birth of the type and recognition of its importance dates from the Richmond meeting of the Institute, in 1901, where three papers dealing in whole or in part with this type of deposit were presented, by J. H. L. Vogt,[4] James F. Kemp,[5] and Waldemar Lindgren.[6] These papers were rightly placed in the Posepny volume on the "Genesis of Ore Deposits" published by the Institute at that time. Since their appearance, the study of contact metamorphism in its relation to ore deposition has had a popularity rivaled only by that of secondary enrichment among the subjects of greatest interest in the field of economic geology. In these investigations Kemp and Lindgren have taken a leading part, knowing this type of deposit better than any other geologists. Consequently, it is an event of unusual interest that the results of independent investigations of the same district by these two men appear almost simultaneously at this time; the fact that there is a certain divergence of opinion as to the genesis of the deposits makes a comparison of their views as expressed in the individual papers all the more interesting. Since it was our good fortune to have these papers at hand during a visit to the Firmeza and Daiquiri mines last November, it is our purpose to present here such a

[4] J. H. L. Vogt: Problems in the Geology of Ore Deposits. *Trans.*, xxxi, pp. 125 to 169 (1902).

[5] James F. Kemp: The Rôle of the Igneous Rocks in the Formation of Veins. *Idem*, pp. 169 to 198.

[6] Waldemar Lindgren: The Character and Genesis of Certain Contact Deposits. *Idem*, pp. 226 to 244.

comparison, in which we are guided by our own observations in the field, and to state the conclusions at which we have arrived.

The interpretation given by Kemp departs from the normal one, and for that reason attention will be directed first to it. In arriving at his conclusions, he had in mind the Iron Springs deposits in Utah, and surely, though he does not mention them, those at Mackay, Idaho.

As Kemp points out, the relations at Iron Springs are the reverse of those in Cuba, since there the largest masses of ore are in the contact zone, whereas the orebodies in the andesite are not of much importance. There is, however, another great difference, in that Leith and Harder,[7] wherever they refer to the deposits in the andesite in their report, distinctly say that they are "true veins or fissure deposits," and hence do not represent a reaction of the magmatic emissions with the constituents of the igneous rock, as do the Cuban deposits, according to Kemp's ideas. It is clear that the Iron Springs ores in andesite represent a mode of ore deposition considerably more remote from that of contact metamorphism than do the Cuban deposits.

Although a deposit of copper ore and not of iron ore, an example that is geologically more analogous is that of the Mackay district in Idaho. These deposits were described in 1907 by Kemp and Gunther,[8] and in 1914 by J. B. Umpleby.[9] According to Kemp and Gunther, there lies between granite and limestone a zone of quartz porphyry, the intrusion of which has produced but little effect on the limestone other than marmorization. Garnetization with associated ore deposition has taken place within the quartz porphyry in the form of pipes and chimneys. Mention is made of the occurrence of blocks of limestone in the igneous rock, but no limestone is associated with the orebodies described. These are conditions very similar to those in Cuba, viz., the presence of orebodies entirely enclosed in the igneous rock and apparently formed at the expense of that rock, and included blocks of limestone in which ore deposition did not take place. There is the additional feature of a large limestone-igneous contact along which little mineralization took place. Commenting on the peculiarity of these conditions, they say: "A new type of orebody is thus afforded." As regards the mechanism of the process by which this garnetization took place in the igneous rock, Kemp says in another paper,[10] "We are forced to the conclusion

[7] C. K. Leith and E. C. Harder: The Iron Ores of the Iron Springs District, Southern Utah. *Bulletin No.* 338, *U. S. Geological Survey* (1908).

[8] J. F. Kemp and C. G. Gunther: The White Knob Copper Deposits, Mackay, Idaho. *Trans.*, xxxviii, pp. 269 to 296 (1907).

[9] Joseph B. Umpleby: The Genesis of the Mackay Copper Deposits, Idaho. *Economic Geology*, vol. ix, No. 4, pp. 307 to 358 (1914).

[10] James Furman Kemp: Ore Deposits at the Contacts of Intrusive Rocks and Limestones; and Their Significance as Regards the General Formation of Veins. *Economic Geology*, vol. ii, No. 1, pp. 1 to 13 (1907).

that the emissions from the deeper parts of the eruptive became charged with lime along the contact and passed upward through the igneous rocks. * * * * Contributions from below of iron oxide and lime have changed it to garnet and other silicates, just as the reverse contributions of silica, iron oxide and alumina to the limestone would lead to the same result."

Umpleby differs from the above geological interpretation only in regarding the quartz porphyry as a marginal phase of the granite and not a separate intrusion, and calls the whole mass granite porphyry. From his descriptions of the ore occurrences, it is seen that mining operations since 1907 have disclosed the fact that most of the apparently isolated garnet masses are in reality in contact with and part of some of the limestone blocks included in the igneous rock, but that at the same time there has been considerable endomorphic transformation of the neighboring igneous rock into garnet rock. In other words, the endomorphism has taken place in exactly the manner described in the above quotation from Kemp, but only in the immediate vicinity of the limestone inclusions. The Mackay district thus affords evidence of an endomorphic transformation of igneous rock resembling the exomorphic transformation of limestone under the action of contact metamorphism, but only at and near the contact with limestone. In the latter respect, there is an important difference between Mackay and the conditions postulated for the Cuban occurrence.

In the preceding comparison of Iron Springs, Mackay, and the Cuban deposits as interpreted by Kemp, it has been shown that, however analogous these occurrences may seem, there are important differences, and that the Cuban case is a distinct exception to the rule that might be formulated for the other two. At Iron Springs there is no endomorphism and no limestone; at Mackay there is endomorphism, but it is associated with engulfed limestone; in Cuba, Kemp postulates endomorphism in the absence of limestone. It is obvious, therefore, that the Cuban case gets no support from previous experience, but must stand on its own merits.

Endomorphism at limestone contacts has not taken place in most contact metamorphic deposits, but examples of it are sufficiently numerous and authentic not to arouse skepticism. To mention but a few additional examples, such a transformation at the limestone-andesite contact is described by Leith and Harder at Iron Springs, and at several localities in Mexico by A. Bergeat[11] and by J. E. Spurr, G. H. Garrey, and Clarence N. Fenner.[12]

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[11] A. Bergeat: Der Granodiorite von Concepcion del Oro im Staate Zacatecas (Mexiko) und seine kontaktbildungungen. *Neues Jahrbuch*, B. B. 28, 1909, pp. 421 to 573.

[12] J. E. Spurr and G. H. Garrey: Ore Deposits of the Velardeña District, Mexico. *Economic Geology*, vol. iii, No. 1, pp. 688 to 725 (1908).

J. E. Spurr, G. H. Garrey, and Clarence N. Fenner: Study of a Contact Metamorphic Ore Deposit. The Dolores Mine at Matehuala, S. L. P., Mexico. *Idem*, vol. vii, No. 1, pp. 444 to 484 (1912).

J. E. Spurr: Theory of Ore Deposition. *Idem*, vol. vii, No. 1, pp. 485 to 492 (1912).

In a recent article in *Economic Geology*,[13] Basil Prescott reviews some facts of occurrence of contact metamorphic deposits and says, "One of the most general characteristics of the contact-metamorphic ore deposits is the development of the ores and silicates at concentrated points, the locus of which is a protrusion of the limestone into the igneous rock, or a mass of limestone included in it." That is, the engulfed blocks of limestone in the Cuban diorite afforded favorable *loci* for the most intense contact metamorphism. What is more natural to assume than that, in some cases, there has been a complete obliteration of the limestone, accompanied by considerable endomorphism, especially in view of the fact that other neighboring examples show the process in various stages of completion through the preservation of more or less limestone? The advocate of such a genesis for the Cuban deposits might deem negative arguments sufficient. The other view requires positive arguments to establish it. Let us see what the considerations are that led Kemp to his conclusions against such *a priori* reasoning.

One argument is a resemblance of polished surfaces of magnetite to a replacement of lath-shaped crystals of feldspar, suggesting the preservation of original rock texture in a replaced diorite. This is advanced merely as a resemblance, and no further emphasis laid on this point. The main emphasis is laid upon the long, narrow shape of the orebodies and their nearly vertical position, and the apparent alignment of the Lola Hill deposits and several others, lying to the south along a great belt of crushing and faulting.

It is to be regretted that both papers deal almost entirely with the Daiquiri occurrences, and that the greatest attention is paid to the Lola Hill deposits, because they are of paramount economic importance. It happens that the Firmeza mines afford exposures of great genetic significance. The Ocaña mine at Firmeza shows unmistakably the weakness of the arguments based on shape and position of the orebodies. At this mine, there are plainly visible several narrow slabs of limestone engulfed in the diorite which are oriented roughly parallel and standing in an upright position. The ore deposit occurs in the diorite between two of these in the same position and with the same shape. One does not hesitate to consider it a replacement of one of the limestone blocks. Though not by any means as large as the ore mass on Lola Hill, this mine has furnished 300,000 tons of ore, and is estimated to contain 100,000 tons more, and is hence not an inconsiderable orebody. Other

[13] Basil Prescott: Some Observations on Contact Metamorphic Deposits. *Economic Geology*, vol. x, No. 1, pp. 55 to 69 (1915).

examples might be cited, as at the West No. 4 mine at Firmeza, to show that the included limestone masses tend to have the shape and position of tilted slabs.

But coming back to Lola Hill itself, where the assumption of blocks of limestone is dismissed as unreasonable in view of the extent of the orebodies, Kemp feels constrained to rely upon the presence of limestone to explain certain phenomena. On the east side of the orebody, at the south end of Lola Hill, there is a streak of garnet rock. In places this is free from ore; at other points considerable magnetite is included in the garnet, and there is more or less of a transition of the garnet rock into the ore itself through increasing quantities of magnetite. This garnet rock is explained as follows: "But in the case of the streak of garnet rock in the San Antonio with admixed calcite but no appreciable iron ore, an included slab or series of slabs of limestone is not unreasonable." Is it not equally reasonable then to extend the idea a little further, and attribute the localization of the entire orebody to such a slab or series of slabs of limestone, supplemented by more or less endomorphism?[14] If we consider more closely the shape and relations of the Lola Hill occurrence, such a view appears all the more probable. It is a nearly vertical ore lens with a length of 2,650 ft., a maximum width of 250 ft., and a vertical dimension of more than 550 ft. The lens pinches rapidly in its lower part, and its termination in depth has been established in the tunnels and inclines driven in connection with the underground mining. The relations of wall rock to ore and the character of the wall rock on the bottom are the same as on the sides of the orebody. If it represents a replacement of diorite along a zone of crushing by ascending magmatic emissions, one certainly would not expect it to terminate so suddenly and at such shallow depth. On the other hand, it does look as though emissions rising through the diorite along some such zone encountered, in the place now occupied by the orebody, rock of an entirely different composition which was far more susceptible to replacement—that is, a block or blocks of limestone.

These arguments bring us to the interpretation of the genesis of the deposits that is advocated by Lindgren and Ross. The fact that in many instances smaller masses of limestone have been but partly metamorphosed, whereas the much larger mass of Lola Hill has been completely replaced, they attribute to the well-known capricious character of contact metamorphism. That the replacement of limestone has been supplemented by extensive endomorphism they appreciate, and speak of the extremely energetic interchange of constituents that took place between the limestone and the igneous rock—an interchange that was so

[14] Mr. Lee Reifsneider, superintendent of the mines, said that there actually is an outcrop of limestone at the south end of Lola Hill, a short distance down the slope from the San Antonio workings, which lack of time prevented us from seeing.

violent as to suggest "that the diorite was fluid when the metamorphism took place." Yet, one gets the impression that they fall short of attributing enough importance to the far-reaching character and particularly to the completeness of this transfer of material, for their description of the endomorphosed diorite makes it nothing more than an altered diorite and not an entirely new product, which is an integral part of the orebody, indistinguishable from the part derived from the limestone.

To sum up our opinions, the Cuban iron ores are contact-metamorphic deposits localized about engulfed blocks of limestone in diorite. In such cases, where there was a limited supply of magmatic emissions, there resulted the contact metamorphism of only a part of the limestone block. Where the supply was ample and the action most intense, not only was the block of limestone completely replaced, but complete endomorphism of the igneous rock on a large scale occurred in the vicinity. Our reasoning leads us to the conclusion that Kemp has unduly emphasized the importance of the endomorphic action and overlooked the function of the limestone in localizing that action; while, on the other hand, Lindgren and Ross have failed to emphasize sufficiently the endomorphism that played so important a rôle in the formation of many of these orebodies. We have the feeling that the true explanation lies in the middle ground between the two views.

II. The Age of the Limestone

Among the many unsolved problems in the geology of the region under discussion, not the least important and interesting is that of the age of the limestone included in the diorite which encloses the orebodies. Most of those who have written on the subject of these iron ores have expressed opinions on the age of this limestone which have been little more than guesses. In 1892, H. Wedding[15] referred it to the horizon of Quenstedt's *Beta* of the upper White Jura, but gave no evidence or statement of the basis upon which the correlation was made. The others have been less specific in their correlation, generally referring it to the Tertiary, and to the older Tertiary. On the geologic map accompanying the Index to the Stratigraphy of North America,[16] only upper Cretaceous and earlier Tertiary are shown, with the exception of a coastal strip of Quaternary, in this part of the Island of Cuba. In the text of this paper (page 643) T. W. Vaughan says: "Strata have been referred to this age (Upper Cretaceous) in the literature on Oriente

[15] H. Wedding: Die Eisenerze der Insel Cuba. *Stahl und Eisen*, vol. xii, No. 12, pp. 545 to 550 (June 15, 1892).

[16] Bailey Willis: Index to the Stratigraphy of North America. *Professional Paper* No. 71, *U. S. Geological Survey* (1912).

Province, but as no fossils have been listed there is doubt as to the extent of the area underlain by the Cretaceous in this portion of the island." No credence, therefore, is placed in Wedding's reference of the limestone to the upper Jurassic. Kemp, and Lindgren and Ross express no opinion on the subject, but Kemp says "fossils would be of extreme interest."

In view of this uncertainty in regard to the age of the limestone, and the interest attaching to its determination on account of its close association with the iron ores, we were quite surprised and highly pleased to find fossils at a small mine known as Barcelona No. 2, which lies a short distance west of the Lola Hill mines. The fossils, which include corals and sponges, were submitted to Dr. T. W. Vaughan, who kindly gave us the following information in regard to them and the age of the rock. The corals belong to the genus *Leptophyllia* (?), a species of which is found in the Cretaceous of Jamaica. Doctor Vaughan says no similar coral is known in any of the Tertiary of the United States or the West Indies. The sponges are also Mesozoic and probably Cretaceous. The age of the limestone therefore is definitely fixed as Mesozoic and probably Cretaceous.

These fossils are of importance as being apparently the first Mesozoic fossils found in this part of the Island, and more particularly as furnishing the first definite information in regard to the age of the limestone associated with the Firmeza and Daiquiri deposits. It was late in the afternoon when we visited this mine and there was little time for collecting. A more careful search than was possible in the limited time at our disposal would undoubtedly reveal other forms, as well as other localities at which the limestone is fossiliferous, and ultimately make possible a still closer correlation.

Additional Data on Origin of Lateritic Iron Ores of Eastern Cuba

BY C. K. LEITH,* PH.D., AND W. J. MEAD,† M.A., MADISON, WIS.

(San Francisco Meeting, September, 1915)

In 1911, we published in the *Transactions* a brief account of the lateritic alterations of serpentine in eastern Cuba, producing the important iron-ore deposits of the Mayari and Moa districts.[1] The special feature of that article was a quantitative treatment of the alterations, based on analyses, to show just what had happened in terms of chemical, mineralogical, and physical changes. Professor Kemp has published an interesting account of these ores, containing additional quantitative observations of his own, tending essentially to confirm the nature of the changes that had been worked out.[2] His discussion suggests to us the desirability of publishing certain additional quantitative data which were not included in our previous article. This seems especially desirable because quantitative measurements of lateritic alterations are rare, and this particular case may be regarded as typical of a fairly wide range of alterations of this kind.

At the time of our examination of Mayari and Moa deposits 29 complete analyses were supplied us by the chemists of the Spanish-American Iron Co., representing a graded series from the unaltered serpentine rock below to the ore at the surface. These analyses are given in the accompanying table.

The mineralogical composition of the ores and rocks is comparatively simple and can be easily calculated from the analyses. In Fig. 1 the mineralogical changes from the serpentine rock below to the lateritic iron ore above, based on the analyses, are expressed in terms of weight, assuming alumina to have remained constant during the alteration.

It is not easy by mere inspection of this table of analyses to determine what the progressive chemical changes have been in development of the ore. In order to bring this out clearly, the analyses have been graphically represented in Fig. 2 in such a fashion as to show the progressive additions and losses of substances during alteration.

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[1] Leith, C. K., and Mead, W. J.: Origin of the Iron Ores of Central and Northeastern Cuba, *Trans.*, xlii, 90 to 102 (1911).

[2] Kemp, J. F.: The Mayari Iron-Ore Deposits, Cuba, *Trans.*, li., 3 to 30 (1915).

Chemical Analyses Showing Alteration of Serpentine Rock to Iron Ore in the Mayari District, Cuba

Analyses supplied by Spanish-American Iron Co.

Depth, Feet	SiO_2	Al_2O_3	Fe_2O_3	Fe	MgO	Cr.	Ni+Co	P.	S.	H_2O+	Total
0- 1	2.58	15.71	66.20	46.37	......	0.92	0.38	0.016	0.12	10.20	96.126
1- 2	2.38	20.81	64.70	45.34	......	0.96	0.33	0.022	0.12	10.63	99.952
2- 3	1.60	17.43	68.40	47.81	......	0.96	0.42	0.018	0.14	9.15	98.118
3- 4	1.42	14.23	68.70	48.09	......	1.04	0.36	0.019	0.16	9.50	95.429
4- 5	1.56	8.47	70.60	49.46	......	1.27	0.61	0.016	0.17	10.14	92.836
5- 6	2.90	10.24	72.35	50.56	......	1.66	0.84	0.016	0.20	10.96	99.166
6- 7	2.20	8.29	72.90	51.00	......	2.19	1.09	0.007	0.19	11.35	98.217
7- 8	2.68	4.92	71.85	50.28	......	2.19	1.15	0.006	0.14	11.57	94.506
8- 9	3.30	7.25	71.55	50.15	......	2.39	1.14	0.006	0.16	12.12	97.916
9-10	2.44	6.91	72.40	50.63	......	2.08	1.21	0.005	0.16	12.35	97.555
10-11	2.42	6.31	71.40	49.94	......	2.00	1.36	0.005	0.14	12.40	96.035
11-12	2.72	7.05	70.55	49.46	......	2.08	1.31	0.004	0.15	12.40	96.264
12-13	2.56	6.77	70.20	49.08	......	1.62	1.37	0.004	0.10	13.50	96.124
13-14	2.52	6.23	70.55	49.46	......	1.85	1.41	0.005	0.14	13.12	95.825
14-15	2.76	6.58	71.85	50.22	......	1.89	1.38	0.007	0.21	12.45	97.127
15-16	2.78	6.53	70.00	48.98	......	2.16	1.33	0.007	0.19	12.35	95.347
16-17	2.98	6.43	69.80	48.84	......	2.19	1.42	0.007	0.19	12.57	95.587
17-18	3.20	5.53	70.45	49.32	......	2.00	1.35	0.007	0.15	12.90	95.587
18-19	3.66	6.51	69.20	48.42	......	2.43	1.34	0.005	0.06	12.73	95.935
19-20	6.84	8.49	63.35	44.32	......	2.51	1.36	0.004	0.08	12.80	95.434
20-21	7.44	5.13	66.55	46.58	......	2.27	1.57	0.003	0.09	12.45	95.503
21-22	8.46	4.99	57.80	40.47	0.00	2.16	1.47	0.006	0.08	12.71	87.676
22-23	11.04	8.38	62.10	43.49	0.00	1.85	1.74	0.002	0.09	14.07	99.272
23-24	15.86	4.70	63.90	44.62	0.00	2.19	1.57	0.003	0.09	11.73	100.043
24-25	17.40	4.00	62.90	40.00	0.50	1.85	1.43	0.003	0.12	11.64	99.843
25-26	22.54	4.57	50.25	35.12	6.49	1.89	1.80	0.002	0.06	13.65	101.252
26-27	28.60	4.18	32.85	23.00	18.23	1.12	1.43	0.003	0.09	13.45	99.953
27-28	35.64	2.33	18.25	12.78	27.35	0.77	1.35	0.001	0.06	14.23	99.981
28-29	39.80	1.39	10.14	7.10	33.69	0.20	0.97	0.001	0.06	13.31	99.561

By means of the "straight-line diagram,"[3] each analysis has been compared in turn with the analysis of the unaltered serpentine rock. Each point platted on the horizontal scale, for a given constituent, represents the number of grams of altered rock or ore necessary to contain the same amount of that constituent as 100 grams of unaltered serpentine rock. For example, only 7 g. of ore at the surface are required to contain the same amount of alumina as 100 g. of serpentine rock, and hence, if alumina has remained constant, 100 g. of serpentine rock produce only 7 g. of ore at the surface. If any particular constituent is assumed to have remained constant, all constituents platted to the left have been relatively increased and those falling to the right have suffered loss.

Starting with the unaltered serpentine rock below, it is seen that magnesia and silica are very rapidly lost, magnesia practically entirely disappearing at a distance of 5 ft. above the serpentine rock. Silica is reduced to a small minimum above 10 ft. above the serpentine rock and

[3] For a detailed description of the straight-line diagram, see *Economic Geology* vol. vii, No. 2, pp. 141 to 144 (Mar., 1912).

continues essentially constant to the surface. Alumina, iron, and chromium rapidly increase in percentage with the loss of the silica and magnesia. Iron reaches constant maximum percentage at about the same depth that silica reaches its minimum percentage. At a depth of about 10 ft. from the surface the alumina curve turns to the left, showing an increased rate of concentration, while the iron and chromium curves turn to the right. The relative increase of alumina as compared with iron and chromium is believed to be due to reduction and solution of iron in the upper portion of the ore and its transportation downward,

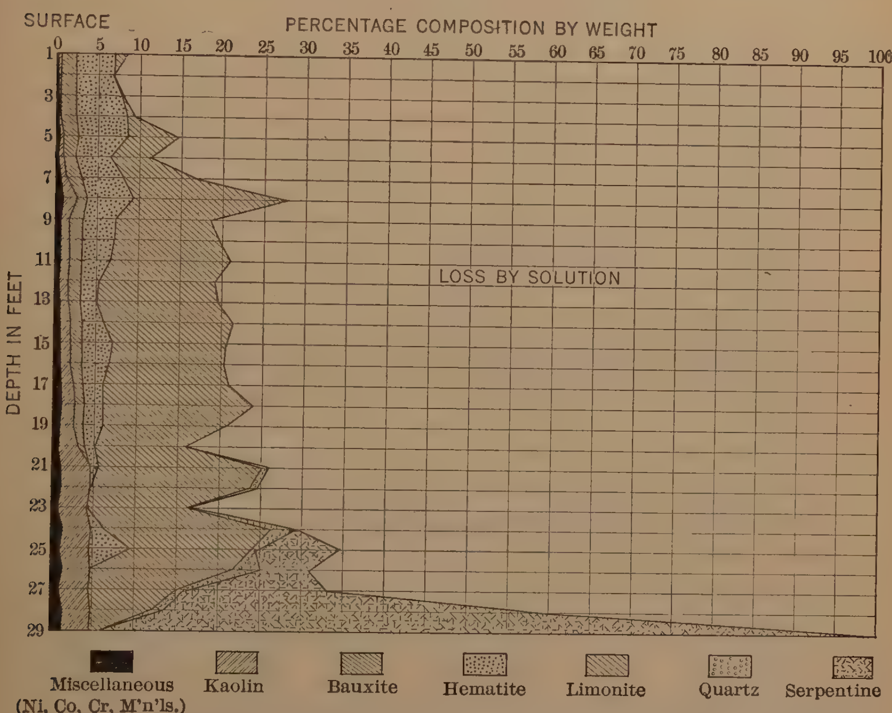

Fig. 1.—Plat of Mineral Changes (based on analyses) in the Lateritic Alterations of Serpentine, Producing the Iron Ores of the Mayari District of Eastern Cuba.

resulting in a downward secondary concentration of iron oxide. Nickel and cobalt are lost relative to iron and alumina, the loss being sharply accelerated at about the same depth at which a sharp acceleration in the relative increase of alumina occurs. The percentage of combined water is comparatively constant but in actual amount combined water is lost relative to iron and alumina.

The marked loss of nickel and cobalt during the formation of the ore suggests the possibility of secondary downward concentration of these metals. Descriptions of the nickel ores of New Caledonia indicate that

they are the result of alteration of peridotic and pyroxenic rocks and of serpentine rocks derived from them, from which the nickel is leached during weathering and carried down and deposited as hydrated silicates

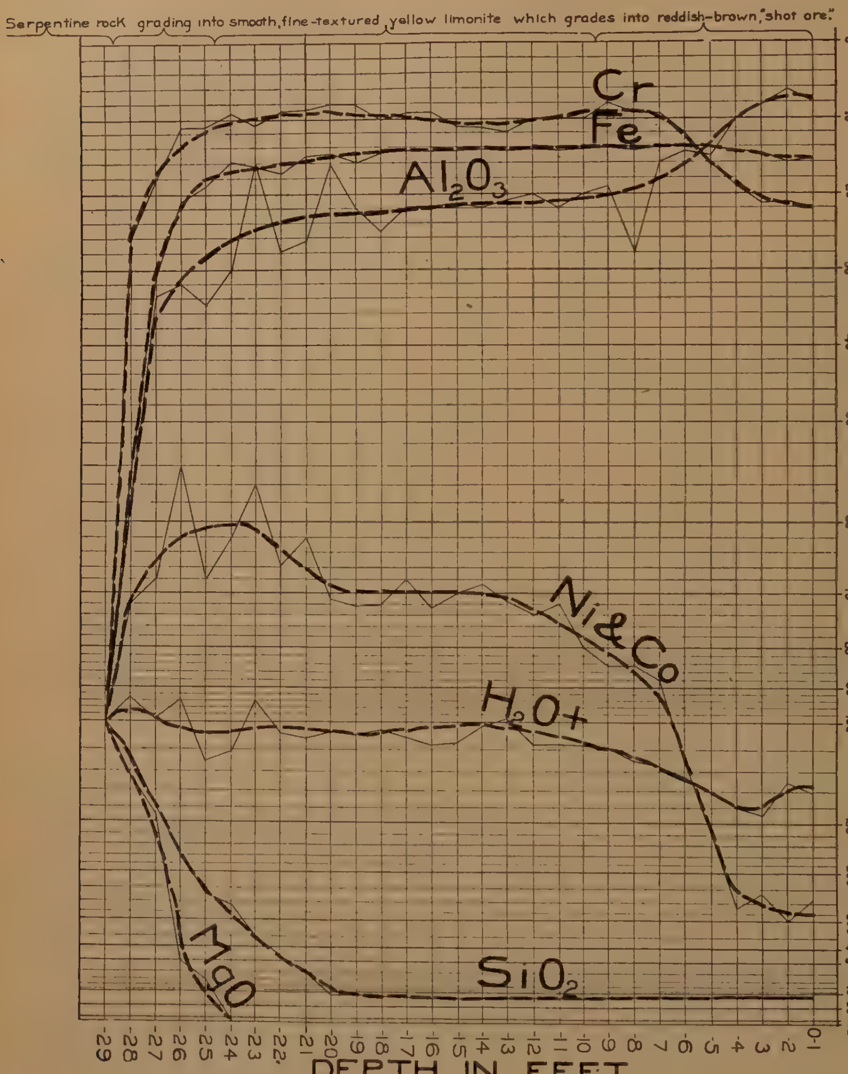

FIG. 2.—PLAT OF CHEMICAL CHANGES IN THE LATERITIC ALTERATION OF SERPENTINE PRODUCING THE IRON ORES OF THE MAYARI DISTRICT OF EASTERN CUBA.

in veins in the underlying unaltered or slightly altered rock. This suggests the possibility of similar occurrence of secondary nickel-bearing minerals in the serpentine rock beneath the Cuban iron-ore deposits.

Conservation of Iron Ore

BY C. K. LEITH, MADISON, WIS.

(New York Meeting, February, 1916)

QUOTING from Dr. Richard T. Ely:[1]

"Conservation, narrowly and strictly considered, means the preservation in unimpaired efficiency of the resources of the earth; or in a condition so nearly unimpaired as the nature of the case, or wise exhaustion, admits. And broadly considered, it means more than the word itself implies, for it naturally includes an examination of methods whereby the natural inheritance of the human race may be improved; and still more broadly considered, and as used in popular discussion, it includes a treatment of the effects of productive conservation methods upon distribution.

"Conservation means a sacrifice of the present generation to the future generations whenever it is carried too far. There is a sharp limit to the economic sacrifice that we may reasonably ask the private person to make for even the present welfare, and the limit is still sharper when we come to consider the interests of future generations."

In a paper on the iron ores of America, presented to the Pan-American Scientific Congress at Washington, I have emphasized the fact that the reserves of iron ores in the two Americas are enormously in excess of the requirements of the present generation, in fact, so large that no shortage is to be anticipated for many generations in the future; that a large part of the value of iron ores is put into them by man's efforts to make them available. This fact is fundamental for the intelligent consideration of policies of conservation as applied to iron ore. As I interpret it, this fact makes it practically unnecessary that the present generation should make any especial sacrifice for future generations in the way of conservation of its iron-ore resources. Even if we were disposed to look forward to a period of 100 or 200 years, with a view to insuring the welfare of the people living at that time, it is doubtful whether any procedure we could now formulate would materially help them in the matter of iron ores, so large are the reserves available. In order that an appeal for conservation may be effective it must obviously favor the welfare of the present or immediately following generations; it must promise benefits to general and individual welfare during a period within the range of comprehension of the average man.

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[1] Richard T. Ely. Economic Aspects of Conservation. Prepared for Pan-American Scientific Congress, Washington, 1915–1916.

Within this more restricted field there is ample opportunity for conservation measures which can be made of practical benefit to the present generation and indirectly help the future. Such measures include a variety of mining and business methods which will tend toward the minimizing of waste of ores and bring the maximum possible return on the investment; in fact, all measures which are taken by any far-seeing business man or corporation looking forward to the maximum return, not for the present year, but for a series of years. The business man measures the relation between the present and the future profit by the interest rate. With this as a basis, he takes such steps as will insure the maximum return to himself or immediate posterity. To make such expenditures as will amount, with compound interest, to an aggregate beyond the probable return in the future, means a sacrifice of the present to the future which does not appeal to him as a business transaction and for which there is no strong demand based on possible shortage for the future. If, for instance, in mining Mesabi iron ore it is found possible to stock certain of the lower grades at such a figure that when compounded at a normal interest rate the money will be returned at some future time, the operator is using a conservation method which is a benefit to himself and immediate posterity. To spend for this purpose an amount so large that the receipts for this stockpiled ore will never catch up with the compound interest on its preliminary investment means sacrifice to the future, which the large reserves of ore do not seem to require. If the miner spends $2 worth of energy and materials to save $1 worth of materials for posterity, he is making a sacrifice too great to be required of an individual. Up to certain limits, it is often cheaper to waste than to save, that is, cheaper when the human factors in the equation are considered as well as the amount of material saved. In such cases wasting may be real conservation when public welfare as a whole is considered. The problem comes down to a question of how much one can afford to spend in order to accomplish a result which will eventuate during a period of years in the future with due regard for the interest rate, which measures the difference between the present and the future.

Such kinds of conservation as above indicated are aimed at in the normal course of enlightened business. The good operator of mines is distinguished from the poor operator to some extent by his ability to think far enough ahead. In planning for the future he often is aided by the demands of the fee owner who is interested in getting the maximum possible returns from the property through the life of the mine. In fact there often develops here a conflict of interest which may even lead to measures of conservation detrimental to the best interests of the operator. With the concentration of control in large corporations has come a natural tendency to think in larger terms and to apply conservation methods which will bring the maximum results through

a period of years representing the probable life of the corporation. In this the interest rate is figured as a matter of course. The well-established large company is not so much at the mercy of immediate requirements as competing smaller interests, but has the means to conduct its operations on the basis of average returns through a long series of years. It does not find it necessary to make a considerable sacrifice of the future for the profit of the immediate years. In this respect, concentration of control has undoubtedly exerted a conserving influence as compared with unrestricted competition working on the principle of *laissez faire.* On the other hand, over-capitalization requiring immediate large returns may be inimical to conservation of resources.

The natural tendency of business to seek a standard of operation determined more or less by the best results of the most far-sighted people will, as a matter of course, involve improvement in conservation methods which will benefit the present generation. The present interest in questions of efficiency and efficiency systems is a step in the direction of conservation, which is leading to an analysis of the factors in the problem and the determination of how far one can wisely go in sacrificing the present for the future in order to get the maximum possible returns. The development of economic theory is leading to a clearer recognition of the factors involved and therefore to plans of operation involving less waste, when the operation as a whole is considered, through a period of years.

Notwithstanding these tendencies, the average of operation of iron mines is far from reaching the standard of efficiency set by some of the more far-sighted and larger companies. There are far too many operations in which the results of the present or next year figure so largely in plans at the expense of future profits. It may be said that such operators are the only losers, but it probably does not require an extended argument to indicate that the general welfare is promoted by the welfare of the maximum number of individuals. There is, therefore, abundant opportunity for the application of educational methods designed to raise the general standard. Innumerable technical societies are disseminating information tending to improve the standard. Government organizations, like the Bureau of Mines, Geological Survey, and various State organizations, tend toward this raising of standard, by making available to all the best results and methods of the few.

The problem of conservation is so complex that, as the writer sees it, it is practically impossible to lay down rules of conservation applicable to iron ore as a whole, to say nothing of all mineral products. There is too great a variety of local conditions. What might be intelligent conservation in one case would amount to confiscation or needless sacrifice in another. It might be possible, for instance, in a given district where it is shown that a certain grade of ore could be conserved for the

future for a certain expenditure, to formulate rules covering this. However, to carry this rule over to another region might require unwarranted sacrifice on the part of the individual in favor of future generations.

The conservation propaganda which has been so vigorously waged in recent years in the United States has, as a rule, not been qualified by the above restrictions. The emphasis has been entirely on the welfare of posterity, implying sacrifice to the present generation, without raising the question whether this sacrifice is in all cases warranted, and without attempt to balance the welfare of the present and the future. There seems to have been little recognition of the principle that measures tending toward the welfare of the present generation are conservational in their effect; are the natural first steps in any conservation plan; are the ones which can be most intelligently formulated; will indirectly benefit future generations, and will logically lead to the development of further measures for the protection of future generations when it becomes clear what balance between present and future welfare should be striven for. The result has been that many mining men, sympathetic with the general idea of conservation, have been repelled by its impracticability when applied to their specific problems. Others carried away with the enthusiasm of the idea have attempted to introduce measures which have become so burdensome and such a handicap in competition that they have been speedily abandoned. A number of cases might be cited of specific and distinct benefits of the conservation movement in initiating specific measures for conservation of ores. The most important result of the movement to date seems to have been to stir people to consider conservation questions and to develop a favorable atmosphere for the introduction of specific measures which on examination may be found to be practicable. It seems to the writer that definite recognition of the fact that conservation involves consideration of the welfare of the present as well as that of the future should be a powerful stimulus to development of effective conservation measures. The changes will be largely accomplished through an individual initiative in attempting to reach the highest business standard. Governmental and educational agencies can supplement effectively these tendencies by spreading the information and in making certain standards compulsory for specific groups of conditions.

For the government or other public bodies to go further, and to formulate rules for conservation which require sacrifice of the individual to distant generations, would necessitate a considerable exercise of public power backed by strong public sentiment, which does not seem to be needed in the case of the iron ores, because of the enormous quantities available, which make it reasonably certain that posterity will be taken care of. The probability is that in the future, as in the past, the individual sacrifice will be limited to sacrifice of the immediate moment for

a gain which can be definitely approximated in the near future. If, however, it becomes desirable to bring public power to bear on the development of the iron ores, with a view to conserving these ores for posterity at a considerable additional expense on the part of present owners, the burden of such sacrifice should be taken by the government and not by the individuals. Quoting from Ely:[2] "When it is possible, and as a general principle, social burdens should be socially diffused and socially borne." Professor Ely[2] argues that at this point public ownership should be considered. "In proportion as the social benefits desired are secured by an increasingly frequent application of public power, the advantages of private property become smaller as contrasted with the advantages of public property."

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[2] *Loc. cit.*

The Iron Mines of the Sierra Menera District of Spain*

EDITED BY A. S. CALLEN, SO. BETHLEHEM, PA.

(New York Meeting, February, 1916)

THESE iron mines of Spain are located on the mountain ridge forming the boundary between the Teruel and Guadalajara provinces, called Sierra Menera. They form a property of 25 mines extending over an area of 1,677 hectares (4,140 acres) situated at a height varying from 1,200 m. (3,940 ft.) to 1,500 m. (4,920 ft.) above sea level. The deposit follows the direction north to south; huge masses of ore are being found both on the slopes and at the summit of the Sierra. The ore occurs among quartzites and magnesite limestone, in some places overlaid by quartzite 15 m. (50 ft.) thick; in some others (and this is more general) the ore is overlaid by earth and boulders of quartzite origin; this layer is about 8 m. (26 ft.) thick. The ore beds occupy the geological horizon of the Superior Trias or Muschelkask, which are here in direct contact with the Silurian quartzites. The lime-stones have been more or less transformed.

There is ample evidence to show that these mines were worked in very ancient times—objects and tools employed in mining by the ancient Romans and Arabs are to be found plentifully in old workings which extend down into the deposits. It has been estimated that more than 10,000,000 tons of ore were mined in the old days in this district and carried to the primitive forges in the vicinity where iron of much renown in Spain was extracted. Remains of more than 20 of these forges are found in the neighborhood of Sierra Menera and are usually situated near waterfalls and forests.

Much investigation has been carried on for the purpose of determining the quantity of ore to be found in these mines, but no exact calculations can be made at present. When the investigation is completed it will surely be found that the deposit contains a much larger quantity than the 100,000,000 tons indicated by the present workings.

The mines are now worked as open quarries and this method can and will be followed for a long time, probably until the deposit is worked out. Figs. 1 and 2 give some idea of the quarries in operation and the reserves, the iron-ore stocks, screening installation and general arrangement.

From these quarries 1,000,000 tons can be mined annually (987,000

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* Through the kindness of Victor de Ysasi some data from Messrs. Sota and Aznar, Bilbao, were received from which this paper was compiled, the editor adding information obtained from *Fairplay*, Nov. 23, 1911.

FIG. 1.—SIERRA MENERA DISTRICT.

tons in 1913) and the production could easily be doubled if the market demanded it.

The ore known as *Sagunto Rubio* is carried directly from the mines to the boats without any treatment. The rest of the production is screened, giving two qualities—one of lump and rubble, the other of fines. The lump and rubble form the class called "screened Menera." This gives an ore of excellent physical condition. The fines, which are under 20 mm., are transformed into nodules or briquets.

The chemical composition of the different classes is shown in the following table:

	Menera, Per Cent.	Sagunto Rubio, Per Cent.	Briquets, Per Cent.	Nodules, Per Cent.
Iron	54.300	53.500	62.000	62.000
Manganese	1.760	0.860	2.040	2.040
Phosphorus	0.025	0.105	0.030	0.030
Sulphur	0.013	0.014	0.015	0.015
Silica	5.160	9.840	6.010	6.010

Railway.—The railway over which the ore is shipped is 210 km. (131 miles) long, 1 m. gage, and extends from the mines to the port at Sagunto, running through the provinces of Teruel, Castellon and Valencia. The country traversed is mountainous and abrupt, which necessitated the building of important bridges and tunnels. The largest bridge is a great viaduct 48 m. (158 ft.) high.

The railway has a large capacity and is well equipped with locomotives and steel cars. The locomotives are of the compound type with six coupled axles with a weight of 14 tons on each. They were built by the North British Locomotive Co., Ltd. The cars are of the automatic hopper discharge type, with two axles, carrying 20 tons each. Both the locomotives and the cars have hand and Westinghouse air brakes. Each train has 22 cars and one locomotive.

Port.—The port is situated in the gulf of Valencia, on the Sagunto beach, between the ports of Valencia and Castellón, 12 miles from the former and 25 from the latter.

There is a breakwater 750 m. (2,460 ft.) long, normal to the shore and another 250 m. (820 ft.) long, perpendicular to the first. Between these a safe anchorage is provided against the levant storms which are the only ones to be feared along these coasts. There are 120 m. still to be built to the second breakwater mentioned. However, without this, the loading and discharging is carried on with great security, even in times of severe storms. At present, two steamers 350 ft. long with a draft of 30 ft. can be loaded simultaneously. When the breakwater is completed it will be possible to work on a greater number of boats simultaneously. (See Fig. 2.)

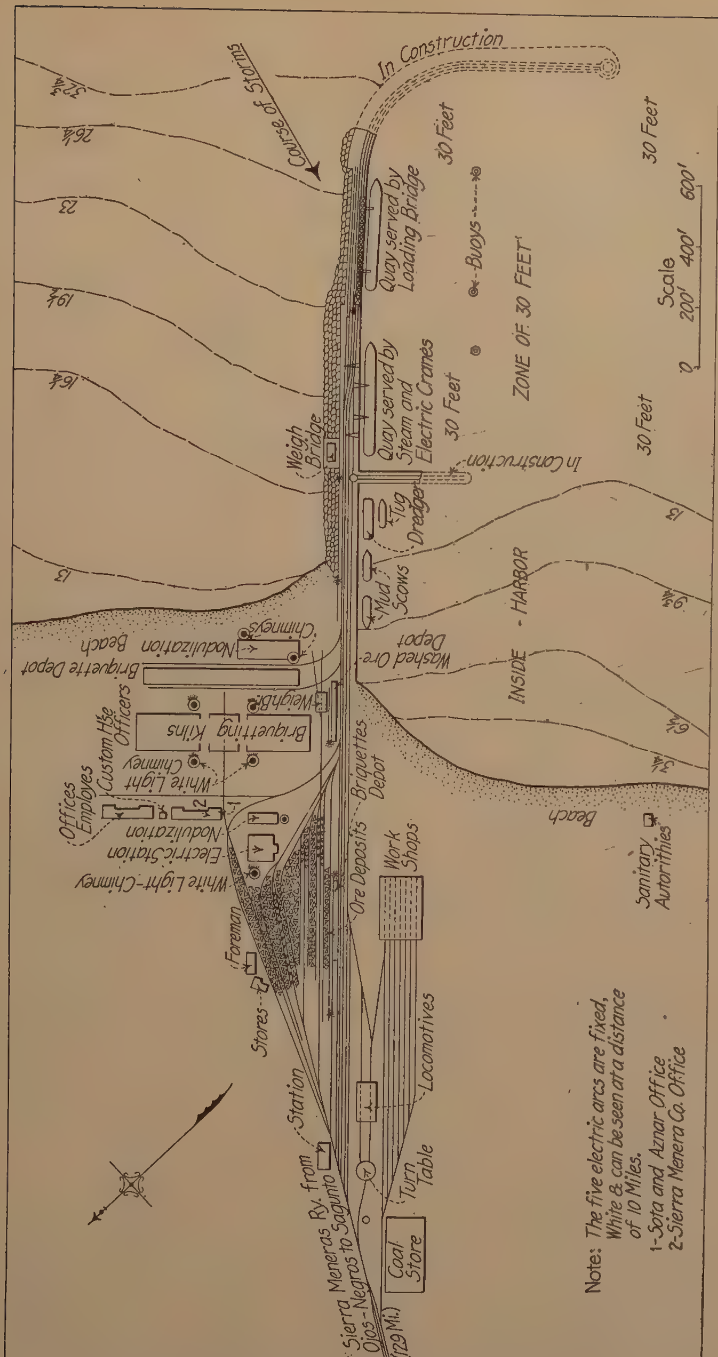

FIG. 2.—GENERAL ARRANGEMENT OF PLANT AT SAGUNTO.

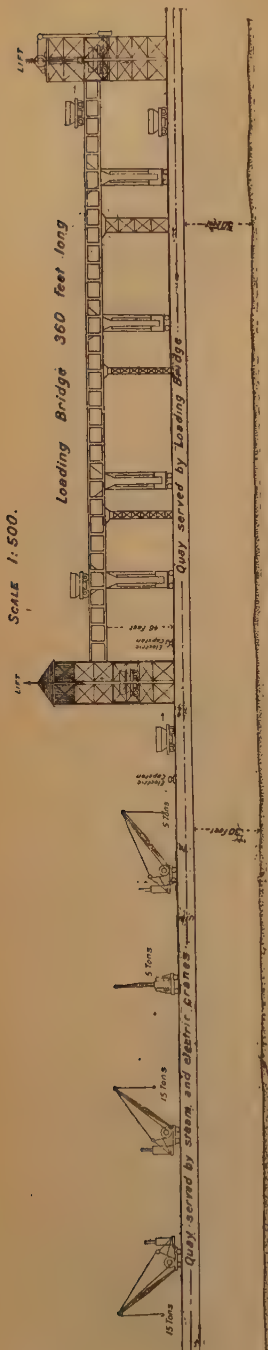

Fig. 3.—Ore Loading Dock at Sagunto.

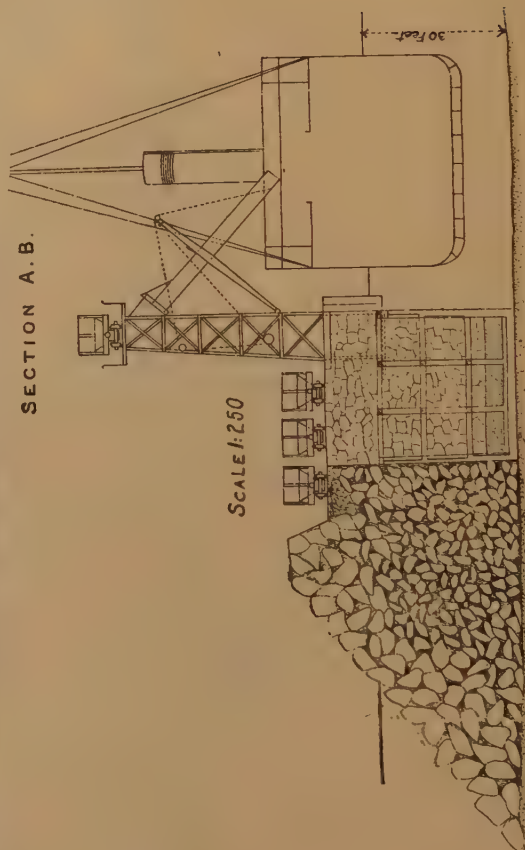

Fig. 4.—Section of Loading Dock, Fig. 3.

The first 1,680 ft. of the first breakwater are of loose blocks; the remainder of the breakwater consists of eight concrete caissons weighing 5,150 tons each. On these caissons a large iron structure, similar to the equipment in the American ore docks, 360 ft. long and 46 ft. high, has been built. At one end of this dock, the railway trucks, 20 tons each, are elevated by a powerful electric lift, to the loading bridge, are run over this bridge and discharged through four chutes into steamers. The empty cars are lowered to the railway line by another elevator at the end of the dock.

The rest of the berthing quay is served by six electric and steam cranes. Two of these cranes are of 15 tons capacity each, and the other four are of 5 tons capacity. (See Figs. 3 and 4.)

The plant for loading and discharging has a capacity of 600 tons per hour. The port has the necessary electric lights for night illumination so that if necessary a night shift may be used. The port is of sufficient depth over a space of 39 acres to maneuver steamers of 30-ft. draft. Dredging is in progress which will increase this greatly. The stock piles at the port are capable of storing 100,000 tons of iron ore. This reduces to a minimum the delay in loading.

On the shore near the loading berths the company has two nodulizing plants, capable of producing 50 and 250 tons of iron-ore nodules, respectively, per day; also a briquetting plant consisting of 17 channel kilns, each producing 65 tons of iron-ore briquets per day, or 1,105 tons of briquets per day per plant.

The port expenses at Sagunto are light; the charge of 10 centimos per ton on the ore loaded includes the use of shore mooring ropes, mooring boat hands, pier pilot and of the quay. In addition, a transport tax is levied of 50 centimos per ton of iron ore shipped from Spain to the United Kingdom or to the Continent; to America the tax is only 20 centimos per ton.

The corporation operating this property is the Compañia Minera de Sierra Minera, domiciled at Bilbao, and Messrs. Sota and Aznar of that city are the managing directors.

The Iron Ores of the Philippine Islands*

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(New York Meeting, February, 1916)

Introduction

Iron-ore deposits in the Philippine Islands became the subject of official record as early as 1664. Undoubtedly iron ore was known and recognized by the Filipinos long before the earliest Spanish records. It is even possible that the Filipino practice of smelting iron ore is, like the native copper smelting, older than the Spanish conquest of the islands. The first records have to do with the magnetite-hematite ores near the towns of Santa Inez and Bosoboso in Rizal Province, from 15 to 20 miles east of Manila. The magnetite-hematite ores of Bulacan were exploited in 1783. Similar ores near the town of Mambulao in Camarines Province yielded a number of specimens to be exhibited in Spain in 1834. With the exception, then, of the lateritic iron ores on the eastern coast of northern Surigao, the true nature of which was not recognized until 1914, the important iron-ore deposits of the Philippines had all been discovered before the islands came under the dominion of the United States.

Indeed, in the Spanish documents relating to the history of iron mining are statements which, if their accuracy be not questioned, leave us in the Philippines today with much less iron ore than the Spaniards found. To quote, for example, a report submitted to the Superior Government in Spain in 1835 by one of its emissaries, Don Lorenzo Calvo:

> "The Philippine Islands, most excellent Sir, possess iron and exhibit it in nearly every known mineralogical combination, and in such abundance that the cordillera of the island of Luzon, from Montufar Point in San Bernardino Straits to Cape Bojeador, is composed of no other mineral than iron, its ranges, of the third order, being in content and structure totally iron of the best quality, the more valuable in that its combinations are such as to facilitate its fusion and to render its reduction to metal simple."

Today, however, it is recognized that the iron-ore deposits in the Eastern Cordillera of Luzon constitute, not a continuous belt, but a series of widely separated deposits, including a half dozen orebodies within a distance of 10 miles in Bulacan Province, a single outcrop at Santa Inez, Rizal Province, farther south, and three deposits around

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† Chief, Division of Mines, Bureau of Science.

the margin of the Mambulao-Paracale gold-mining district in Camarines Province, 100 miles farther to the east-southeast. Other unimportant

Fig. 1.

occurrences of magnetite-hematite ores are known, but the foregoing, together with the lateritic ores in Surigao Province, Mindanao, contain the economically important iron-ore reserves of the Philippines.[1]

Magnetite-Hematite Ores

General

The deposits of magnetite-hematite ores are found in the lower part near the base of the Miocene sedimentaries and along the contact of this series with the older igneous-complex upon which it rests. Some of the orebodies are entirely within the older igneous rocks close to the contact. Intrusive rocks are associated with each of the orebodies and penetrate the sedimentaries, the older igneous rocks, and even the orebodies at places. The igneous-complex includes deep-seated, intrusive, and extrusive types; in Bulacan, intrusive diorite and andesite, andesite flows, agglomerates, and tuffs; in Camarines, peridotite and pyroxenite. Both in Bulacan and in Camarines, also, there are areas of hornblende-granite which are older than the sedimentaries. In Camarines the granite has been rendered gneissic and the peridotite schistose, probably by regional dynamism. The sedimentary column is of varying thickness, but usually exceeds 3,000 ft.; at the base are breccias, conglomerates, and volcanic tuffs inclosing crystalline limestone; these are overlain by shale, subordinate sandstone, and tuff-sandstone, with two other limestone horizons in the upper part of the series.

Although outcrops of intrusive rocks are abundant near the orebodies, the exposures are small in area, discontinuous, and difficult to delineate. The predominant type of intrusive is felsitic, porphyritic, or holocrystalline in texture and subsiliceous in composition; the rock is dark colored and contains plagioclase feldspar and hornblende or, less commonly, pyroxene as essential minerals. Near one of the orebodies in Bulacan a siliceous rock is encountered in apparently intrusive relations; in this rock, quartz and feldspar phenocrysts, much cracked and with corroded or ragged outlines, are embedded in a microcrystalline groundmass of

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[1] The following references on Philippine iron ores contain descriptions of the various deposits:

Dr. F. Rinne: *Zeitschrift für Praktische Geologie*, vol. x, p. 113 (1902).

Hiram Dryer McCaskey: A Geological Reconnaissance of the Iron Ores of Bulacan, *Bulletin of the Philippine Islands Mining Bureau*, No. 3 (1903).

G. I. Adams: Geological Reconnaissance of Southwestern Luzon, *Philippine Journal of Science*, vol. v, p. 106 (1910); contains notes on the Santa Inez, Rizal, ore.

F. A. Dalburg and Wallace E. Pratt: The Iron Ores of Bulacan Province, P. I., *Idem*, vol. iii, Sec. A, p. 201 (1914).

F. T. Eddingfield: A Microscopic Study of the Bulacan Iron Ores, *Idem*, vol. iii, Sec. A, p. 263 (1914).

Wallace E. Pratt and Victor E. Lednicky: Iron Ore in Surigao Province, P. I., *Idem*, vol. v, Sec. A, p. 335 (1915).

Wallace E. Pratt: Iron Ore on Calambayanga Island, Mambulao, Camarines Province, P. I., *Idem*, vol. v, Sec. A, p. 323 (1915).

quartz and feldspar. The feldspar, both in the phenocrysts and in the groundmass, is principally orthoclase, but plagioclase feldspar is also present.

The walls of the ore deposits in Bulacan, for a distance equal, roughly, to the thickness of the orebody, consist almost invariably of an altered green rock of uncertain composition. So constant is the association of this wall rock with the ore that the relation is recognized by the native miners, who designate the greenstone as *camisa de bacal* (shirt or cloak of the iron ore). The wall rock is made up principally of amphibole and pyroxene with varying proportions of chlorite and epidote and often some plagioclase feldspar. Frequently the amphibole is fibrous and at many places hand specimens may be secured which consist entirely of either finely or coarsely fibrous amphibole. Eddingfield[2] identified the finely fibrous mineral in several specimens as tremolite. Adjacent to the ore the wall rock contains magnetite in grains and crystals of pyrite, disseminated and in veinlets with quartz. The apparent similarity of the wall rock at the Bulacan deposits to the "skarn" associated with the Scandinavian iron ores has been suggested in descriptions of the Bulacan ores.

The orebodies usually conform in strike and dip with the sedimentary formation and consist in some cases of replacements of sedimentary beds, especially limestone. Igneous rocks have also been replaced, however, and open fissures and cavities of irregular shape have been filled with ore. The surface dimensions of the outcrops are usually not great, the longest continuous exposure being something over 1,500 ft. in length. The orebodies have not been explored beneath the surface and some of the outcrops consist simply of blocks of ore scattered over the surface and embedded in clay.

The ore consists of magnetite and hematite in intimate mixture, but in proportions which vary at different outcrops. Magnetite, which is generally predominant, occurs as massive grains and the magnetite ores are usually soft. Hematite may be present only in negligible proportions or may constitute an ore to the practical exclusion of the magnetite. The hematite ores are harder than the magnetite and form resistant outcrops. The hematite is generally massive, but specularite occurs sparingly in nearly all of the deposits. Quartz is an abundant gangue mineral in the leaner ores and near the walls of the orebodies. It occurs in the interstices between the grains of magnetite and hematite. Eddingfield's microscopic work revealed needles of hematite penetrating the quartz in radiating groups and led him to the conclusion that the two minerals crystallized from the same solution. Quartz occurs, also, in secondary relations cutting the ore in small veins. A great deal of the country in the vicinity of the orebodies, likewise, is thoroughly silicified.

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[2] *Loc. cit.*, p. 266.

Amphibole, chlorite and epidote are found in the ore near the walls. Pyrite occurs as grains and crystals in the ore as a primary constituent (according to Eddingfield) and as veinlets of secondary origin in the ore and walls. Pyrite is so abundant at places as to be objectionable in smelting but much of the ore, also, is free from pyrite. Rarely, chalcopyrite is found with the pyrite.

The geology of the iron ores will be made clearer, perhaps, by the following brief descriptions of the principal orebodies.

Descriptions of the Principal Deposits

The largest outcrop of iron ore in Bulacan Province is on the western flank of the Eastern Cordillera about 50 miles north of Manila, in a mountainous, inaccessible, and uninhabited region to which the name Camaching is applied. Camaching, in common with the other ore deposits in Bulacan, is in a heavily forested, broken country which can be entered only along narrow mountain trails. The nearest towns are 10 miles distant, while the railroad is 5 miles farther away. The Camaching deposit is at the head of a small river on a steep westward slope at an elevation of about 1,000 ft. In approaching it from the west along the river, the visitor traverses an extensive series of bedded tuffs and flows which represent the lower part of the Miocene sedimentaries. These beds strike north 20° east and dip to the westward at an angle of about 45°. The iron ore is encountered between the usual greenstone walls, previously described, near the base of the sedimentaries and the deposit conforms with the sedimentaries in strike and dip. The altered wall rock overlying the ore grades into fragmental bedded rocks containing a large proportion of volcanic tuff. Pieces of float ore downstream from the outcrop preserve the texture of the fragmental rock which the ore has replaced. In the wall of a stream crossing the outcrop an andesite dike, which follows the strike of the orebody, is exposed. The dike is about 15 ft. wide and is inclosed on both sides by ore. Similar dikes are encountered in the bedded rocks overlying the ore. Rounded blocks of crystalline limestone are included in the ore at places and upstream (stratigraphically beneath the orebody) is white crystalline limestone cut by small veins of magnetite. Blocks of calcareous, red hematite, evidently replacements of limestone, are found nearby. Farther up the slope sandy shale and tuff, which underlie the limestone, outcrop and, finally, at the top of the ridge are andesite flows which extend to the eastward for several miles.

The orebody as revealed by its outcrop is 1,500 ft. or more in length and from 60 to 200 ft. wide in different exposures. The ore is principally magnetite, soft and massive. Quartz and pyrite occur in the ore and in the walls in the usual relations as previously set forth.

Another deposit of iron ore is found at Montamorong, about 4 miles south-southwest of Camaching. The orebody at this place lies on the eastern margin of a lens-shaped area of granite, elongated in a north-south direction but not exposed as far north as Camaching. The Miocene sedimentary series rests directly upon this granite along its western border and is found in patches on the eastern side of the granite, also. Detritus from the granite can be found in the lower beds of the sedimentaries, a condition which establishes the relative age of the two formations. For several miles to the east of the orebody at Montamorong the rocks are so thoroughly silicified as to be classified with difficulty but probably are tuffs and effusives of greater age than the ore deposit. In the vicinity of the orebody there are a number of exposures of unaltered subsiliceous rocks which are believed to represent intrusions, although the contact relations are not clear. The ore deposit, as revealed by a shallow pit, is a tabular body of magnetite from 3 to 7 ft. in thickness. The strike appears to be northwest and the pitch northeast, at an angle of 45°. The ore is soft and massive and contains quartz and considerable pyrite. The greenstone walls, likewise, carry much quartz and pyrite.

About 6 miles south-southwest of Camaching and in line with the strike of the orebody at that place, there are three outcrops of iron ore within a distance of ½ mile, along a north-northeast line. These outcrops are exposed in the cañons of small streams and while the ore cannot be traced across the intervening ridges and may not persist from one orebody to another, yet the deposits are probably on the same structural line and related in origin. The outcrops are near the eastern margin of the granite area and are themselves flanked on the east by discontinuous exposures of sedimentaries. The sedimentary beds consist of schistose mottled limestone, schistose black shale, altered green tuff, and sandstone. The strike is uniformly north 15° east, parallel to the strike of the orebodies, and the dips are steep but are reversed in direction in different exposures. There are intrusive rocks in the immediate vicinity cutting both the granite and the sedimentaries. The quartz-bearing intrusive previously described is exposed between the middle and southern orebodies. Intrusives of subsiliceous composition are also present but at no place are the relations of the intrusives to the ore deposits clear.

The northernmost of the three outcrops is on the Constancia mining claim. The outcrop strikes north 15° east and the ore is apparently about 4 ft. thick, although it is stated that former exploration revealed a thickness of about 15 ft. The ore is magnetitie with considerable pyrite but very little quartz. In the hanging wall magnetitie occurs in closely disseminated grains enmeshed in finely fibrous amphibole.

The Hison outcrop, 1,600 ft. south-southwest of Constancia, is

opened by a shallow pit, 15 ft. wide and 10 ft. deep. The floor of the pit is entirely in magnetite and magnetite is exposed in the eastern and southern walls; blocks of magnetite and hematite, likewise, cover the hillside to the south. On the west, the pit is bounded by one wall of the deposit, striking north 20° east. The green, altered wall rock is particularly well exposed here, and the ore adjacent to the wall is contaminated slightly with amphibole and chlorite. The ore is free from pyrite and contains but little quartz although both of these minerals are found in the walls; in one wall is a small vein of pyritiferous quartz which strikes north 20° east.

About 1,000 ft. south-southwest of Hison is the third outcrop, called Santa Lutgarda. It has a tabular form, 15 ft. in thickness, and strikes north 15° east with a pitch of 60° to the west. The altered material of the foot wall grades into greenish tuff. The ore is hematite with but very little magnetite and contains more quartz than is present at the adjacent outcrops. In lean portions of the ore grains of massive hematite are set closely in a quartz groundmass. The hematite ore from Santa Lutgarda can be distinguished by a slight reddish tinge in contrast with the blue-black color of the magnetite ores. A small proportion of the Santa Lutgarda hematite is of the specular variety.

At Santol, a little more than 2 miles southwest of Santa Lutgarda, large blocks of hematite are encountered over an area 1,000 ft. long and 500 ft. wide on the lower slope of a hill. The blocks are embedded in residual clay and while some of them weigh, perhaps, 100 tons, no ore in place is to be seen. Santol is at the southern end of the granite exposure and the rocks beneath the hematite-strewn area are granite and andesitic intrusives, effusives, and tuffs in undetermined relation. Both on the east and west, Santol Hill is flanked by sedimentary rocks in which crystalline limestone is prominent. Blocks of the same limestone are found, also, mingled with the iron ore and lying just above the iron ore on the slope, but the direct replacement of limestone by iron ore, as observed at Camaching, is not evident at Santol. The ore is hematite, generally massive, with but little magnetite, and carries pyrite and much quartz. The blocks of ore appear to be arranged along a northeast line and iron-stained quartz can be traced to the northeast and to the southwest beyond the ends of the ore-strewn belt. The sedimentaries on either side of Santol strike north and dip west.

At Tumotulo, within 2 miles to the southwest of Santol, there is an insignificant quantity of iron ore which is interesting because, unlike the other Bulacan ores, it is titaniferous. The ore is found on the slope of a hill entirely within the sedimentary area. The hill is made up of shales and is capped by limestone. Small pieces of ore, a massive magnetite-hematite mixture, together with pieces of porphyritic andesite are encountered on the surface immediately downhill from the limestone.

Neither the ore nor the igneous rock is revealed in place. As shown by the analyses in Table I, the Tumotulo ore contains more than 9 per cent. of titanium.

There are several other places in Bulacan where small quantities of iron ore are obtained for smelting, but these minor deposits need not be described here.

The Santa Inez iron-ore deposit in Rizal Province, east of Manila, is in an area of sedimentary rocks intruded by andesite. The orebody is exposed in a small stream on the face of one of these intrusions. It is not apparent whether or not the ore penetrates the andesite. Crystalline limestone occurs farther upstream lying on the andesite and small veins of iron ore in limestone are to be observed in some of the loose blocks. The ore is principally hematite and is pyritiferous; chalcopyrite accompanies the pyrite in places. Blocks of hematite of many tons weight lie in the stream below the outcrop.

In Camarines Province iron ore is found in sedimentary rocks bordering an area of older peridotite and hornblende-granite. Associated with the iron-ore deposits in the base of the sedimentaries are numerous small intrusions of andesite, diorite, and gabbro. The largest outcrops of ore are on Calambayanga Island in Mambulao Bay and on the adjacent mainland. Mambulao Bay is a well-known harbor and there is deep water alongside Calambayanga Island so that the ore could be loaded on ships at a minimum expense. The rocks inclosing the orebodies on the island and on the mainland near the island are steeply dipping, thin-bedded shales, sandstones, clastics, and conglomerates, with fragmentary exposures of crystalline limestone. The general strike of the beds and the trend of the orebody are north-northeast. Calambayanga Island is elongated in the same direction and a peninsula extends to the north-northeast from the mainland to within 2,000 ft. of the southern end of the island.

The island is ¾ mile long and half as much in width and blocks of iron ore, the average dimensions of which vary from 2 to 15 ft., are scattered profusely over its western half, from sea level up to an elevation of more than 200 ft. Along the beach the ore blocks are piled one on another but farther up on the slopes the individual blocks are embedded in the surface clay. Extending into the island from its southern extremity is a great body of quartz, 200 ft. wide and 75 ft. high. The quartz is iron-stained and intersecting veinlets of hematite cut through it in many places, while the sedimentary rocks adjacent to it are silicified. Toward the center of the island the quartz outcrop becomes concealed by a mantle of clay and on the northeastern point of the island where it should appear, if it continued so far, large blocks of iron ore are piled high, one on another, over a belt 100 ft., or more, wide. Shales and sand-

stones, dipping at high angles, outcrop on either side of this ore but several yards to the south along the western shore of the island is a conspicuous dike, 20 ft. in width. The dike rock is holocrystalline in texture and consists essentially of plagioclase feldspar and decomposing pyroxene with accessory magnetite. South of the dike, which is vertical and strikes north 60° west, are volcanic tuff and breccia with blocks of ore again in great profusion 200 ft. farther south along the beach.

The ore is finely porous hematite with traces of fresh pyrite and quartz. It is moderately soft and the natural surfaces of the blocks are extremely pitted and irregular. Outcrops similar to those on the island occur on the point of the mainland nearby and reappear at intervals for a distance of more than a mile inland.

There is a deposit of magnetite at the village of Batobolani (the native word for magic stone, that is, magnetite) 7 miles southeast of Calambayanga Island. The ore occurs in large blocks embedded in yellowish-brown residual clay and covers an area about 1,200 ft. long by 600 ft. wide on the side of a hill. Dark-colored crystalline limestone and a subsiliceous holocrystalline rock which Dr. Rinne[3] determined as hornblende-diorite, are found in loose pieces with the ore. The country at Batobolani is made up of intruded sedimentaries near the margin of the older peridotite-granite area.

A deposit of hematite, much of which is specular, of the same general character as the Batobolani occurrence but covering a smaller area, is found on the eastern border of the peridotite-granite formation about 12 miles east-southeast of Calambayanga Island.

Quality of the Magnetite-Hematite Ores

The chemical analyses in Table I afford an idea of the quality of the magnetite-hematite ores. These and other analyses show the total iron content to be usually more than 60 per cent. Silica is low in proportion to alumina except where the ore is quartzose, and magnesium oxide is noticeably high relatively to calcium oxide in several ores. Sulphur and phosphorus are usually not objectionably high. Attention has already been directed to the one titaniferous ore from Tumotulo, in Bulacan Province. Traces of cobalt have been detected in pyritiferous portions of some of the ores from Bulacan and it is a common experience to find fragments of cobalt-blue slag on the smelter dumps in that province. The analyses of Bulacan ore are upon samples taken from charges for the native smelters in several cases and probably show slightly more iron and less sulphur than the average ore contains.

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[3] *Loc. cit.*, p. 117.

TABLE I.—*Analyses of Magnetite and Hematite Ores from the Philippines*

Constituent	1	2	3	4	5	6	7
Moisture (100°C.)	0.25	0.05	....	0.07	0.27	1.21	....
Silica	5.02	4.66	43.76	6.84	3.32	9.52	1.02
Alumina	4.80	3.68	2.25	4.13	2.36	5.43	1.31
Ferric oxide	66.41	62.76	40.45	59.09	89.56	65.13	97.35
Ferrous oxide	20.64	27.32	13.50	27.68	4.13	7.89	....
Lime	0.35	0.21	nil	0.20	0.16	0.38	....
Magnesia	0.74	1.14	....	1.20	trace	0.58	....
Manganese oxide	0.24	0.08	....	0.15	0.05	0.38	0.11
Titanium oxide	0.23	trace	....	0.20	trace	9.31	....
Phosphorus pentoxide	0.12	0.14	....	0.13	0.08	0.37	....
Sulphur	0.02	0.21	....	0.21	0.06	0.01	....
Carbon dioxide	1.10	0.30	....	0.30	0.18	0.53	....
Total	100.32	100.50	99.96	100.13	99.90	99.53	99.79
Iron, metallic	62.54	65.17	38.80	61.71	65.91	51.72	64.14
Phosphorus	0.052	0.061	....	0.057	0.035	0.161	0.001

1. Camaching ore, smelter charge.
2. Hison ore, running sample across face 23 ft. long.
3. Quartzose ore from Hison.
4. Montamorong ore, smelter charge.
5. Santol ore, composite sample of spalls from many blocks.
6. Tumotulo ore, smelter charge.
7. Calambayanga ore, average composition of a 500-lb. sample.

Analyses 1 to 6, inclusive, by Forrest B. Beyer, formerly chemist Bureau of Science. Analysis 7 by T. Darjuan, chemist, Bureau of Science.

Genesis of the Magnetite-Hematite Ores

All the different deposits of magnetite and of hematite and of mixtures of these two minerals that have been described are believed to be related in origin. The features common to all of them are sufficient in number to justify this conclusion. The intimate association of the ore minerals with quartz and pyrite; the obvious replacement by ore of adjacent rocks, especially limestone, but, also, other types both sedimentary and igneous; the presence of intrusions near the orebodies; the character of the walls of the deposits; taken together, these observations point to the conclusion that the orebodies are the product of metamorphism resulting from intrusion, that is, metamorphism on or near the contact of intrusives with the intruded rocks. The presence of some of the typically contact minerals, like garnet, for instance, was nowhere detected but it may be suggested that, the intrusives being evidently limited in size, metamorphism may not have been extreme enough to have produced typical contact-metamorphic results. It is possible that hematite may have resulted at places from the oxidation of original

magnetite but it appears that hematite is also an original constituent with quartz in some of the ores. Pyrite is also a primary mineral according to Eddingfield.

Adams[4] believes that the mineralization at Santa Inez resulted from the intrusion of andesite into sedimentary rocks. Rinne[5] explained the Batobolani magnetite in Camarines as a product of contact metamorphism. C. M. Weld[6] has described magnetite-hematite ores near Hongkong which are like the Philippine deposits in many respects and which he attributes to contact metamorphism resulting from the intrusion of the Hongkong granite into older sedimentaries. While the Hongkong granite is known to be of wide distribution and the Bulacan granite is similar to it in character, so that the correlation of the two rocks is not impossible, the Bulacan ores can not have resulted from the intrusion of the granite there into the sedimentaries, inasmuch as the granite is clearly older than the beds which have been replaced in part by iron ore. The intrusions involved in the mineralization in Bulacan are believed to be represented by the dike rocks, the presence of which has been noted. Dr. James F. Kemp has emphasized[7] the striking similarity of the Bulacan iron ores as described by Dalburg and Pratt to contact-metamorphic iron ores at Daiquiri, Cuba.

Quantity of Magnetite-Hematite Ores

It will be obvious from what has been said that no accurate estimates of quantity can be made for the magnetite-hematite ores. Dalburg and Pratt,[8] basing their estimate solely on outcrop dimensions and assuming that the orebody would continue in depth a distance equal to its least surface dimension, obtained 1,100,000 tons for the Camaching deposit. The other Bulacan deposits probably aggregate at least 100,000 tons. While the assumption as to the persistence of the ore in depth is conservative in view of the character of the mineralization believed to have caused these ore deposits, yet the data available are insufficient to make the estimates reliable. It is possible that the ore will contain a greater proportion of pyrite in depth and, consequently, should not be included in estimates of tonnage, but there is no indication on the surface that pyrite actually increases in depth. Removal of pyrite by oxidation near the surface, which would result in a proportionally greater pyrite content below the oxidized zone, does not appear to have taken place to any important extent. For the deposits of ore in

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[4] *Loc. cit.*, p. 106.

[5] *Loc. cit.*, p. 113.

[6] *Trans.* l, 236 to 245 (1914).

[7] *Trans.*, This volume, p. 38.

[8] *Loc. cit.*, p. 235.

Rizal and Camarines no estimate of quantity can be offered. The Calambayanga deposit appears to be large; probably 100,000 tons are represented in the blocks of ore on the surface.

Lateritic Iron Ores

General

On the eastern coast of Surigao Province in Mindanao there is an area of about 40 square miles, bordering the sea for a distance of 10

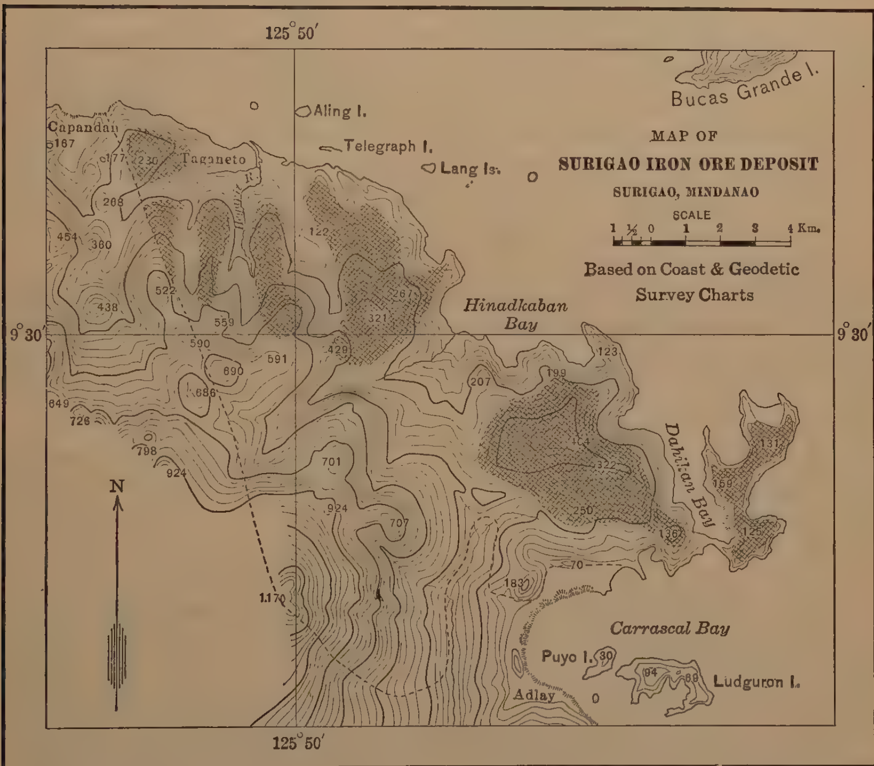

Fig. 2.—Topographic Map of Iron-Ore Deposit in Surigao. Cross-hatched Areas Best Parts of Deposit. Broken Line shows Approximately Interior Limit of Deposit. Elevations in Meters.

miles, which is conspicuously barren of vegetation and is covered with a bright red soil. This condition is the more notable in that the surrounding country is heavily forested and the singular barren appearance of this one section of the coast line has been a subject of interest for years to those who knew of it. But the region is sparsely inhabited and but few boats pass along that coast; consequently it was not until the year 1914

when H. F. Cameron, an engineer who is familiar with the iron ores of the Nipe Bay region in Cuba, made an inspection trip through Surigao that the "Red Hills," as the area in question has been designated on the Coast and Geodetic charts, were found to be covered with laterite rich enough in iron to constitute an ore. The first analyses of the ore were made in the Bureau of Science at Manila and a preliminary survey of the deposit has been made by the Division of Mines, Bureau of Science.

The laterite mantles a dissected plateau ranging up to 1,500 ft. in elevation and terminating along the coast in sea cliffs and steep slopes. This part of Mindanao receives a rainfall that averages about 10 ft. annually. Numerous small streams drain the area covered by laterite and, owing to the copious run-off have eroded deep cañons along their courses. The iron-ore deposit is thus separated into a number of small tracts by precipitous valleys on the walls of which no ore remains (Fig. 2). A peculiarity of the scant, shrub-like vegetation encountered over the laterite region is that most of the plants are common only to high altitudes in the Philippines, several of them not being found elsewhere below an elevation of 5,000 ft., whereas, here, they grow within a few hundred feet of sea level.

While there are no regular harbors near the ore deposit, Dahikan Bay, at its southern extremity, is almost landlocked and is perfectly protected in all seasons; its entrance is narrow but the water is deep, ranging from 18 to 28 fathoms within 300 ft. of the beach, and the inner bay is abundantly large. Two of the streams that reach the coast within the area covered by iron ore are large enough to afford considerable power, although it is not possible to present a definite estimate in this connection.

Character of the Iron Ores

The ore deposit and the ore itself are strikingly similar to the lateritic iron ores of the Mayari district in Cuba, as described by Kemp[9] and by Leith and Mead[10] before the Institute.

The Philippine laterite, or iron ore, is a surface blanket of residual clay varying in thickness up to 60 ft. and resting upon the parent rock from which it has been derived by tropical weathering processes. The surface and upper part of this mantle of hydrous iron oxides are brilliant reddish brown in color but at a depth of a few feet the color fades to a yellowish brown that persists downward to a point where the decomposition of the underlying rock is incomplete. In this zone of incomplete decomposition, which extends upward only 2 or 3 ft. from the hard, unchanged rock, the

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[9] James F. Kemp: The Mayari Iron-ore Deposits, Cuba, *Trans.* li, pp. 3 to 30 (1915).

[10] C. K. Leith and W. J. Mead: Origin of the Iron Ores of Central and Northeastern Cuba, *Trans.* xlii, pp. 90 to 102 (1911).

color is pale green or almost white. The ore throughout its thickness is a spongy or mealy clay, but over its upper surface there are abundant small red concretions, together with occasional rusty porous fragments or crusts of the parent rock.

The parent rock, wherever exposed within the limits of the ore deposit, is essentially serpentine. Along the beach bordering the deposit there are many outcrops which uniformly represent rocks of the subsiliceous class although the textures vary from preponderating holocrystalline to porphyritic, or even felsitic. No petrographic studies have been made of the specimens collected from these outcrops, but the most prominent rock may be classed megascopically as peridotite; probably, also, the porphyries and felsites, which are generally less completely decomposed than the holocrystalline rocks, represent small intrusive bodies.

Miocene sedimentaries, tuff-sandstones and limestone, abut upon the area of igneous rock covered by the ore and the contact between the two formations marks the interior limit of the ore deposit.

Composition of the Lateritic Iron Ore

In composition the Surigao iron ore appears to be slightly inferior to the ores of the same character in Cuba. The Surigao deposit was sampled by drilling, the material from each 10 ft. of hole constituting a separate sample. Thirty-one analyses made in the Bureau of Science upon material selected as representative of a total of 183 samples, taken at various depths from 89 different drill holes distributed throughout the ore deposit, show the average dry ore to contain 12.87 per cent. water and 47.40 per cent. metallic iron. Ore of this character would necessarily be sintered or nodulized before smelting and through the sintering process it would lose part of its water. Kemp[11] observes that the Cuban ore after being sintered contains from 3 to 3.5 per cent. of water. If the water in the Surigao ore is reduced to 3.5 per cent. by sintering prior to blast-furnace treatment the total iron content becomes 52.5 per cent., a figure which may be compared directly with 55.5 per cent. for the average sintered Cuban ore. If two conspicuously poor samples, both of which are from the bottoms of drill holes and are probably, therefore, contaminated with the underlying rock, be excluded from the 31 samples chosen as representative, the average iron content of Surigao ore, containing 3.5 per cent. of water, is 53.9 per cent.

The results of the sampling tests indicate that the Surigao ore is remarkably uniform in composition throughout the deposit. Of the 31 samples analyzed, 28 show more than 45 per cent., 24 show more than 50 per cent., 12 show more than 55 per cent., and three show more than

[11] James F. Kemp: *loc. cit.*, p. 5.

60 per cent. of metallic iron on the basis of 3.5 per cent. water content. Of the three samples containing less than 45 per cent. of iron on this basis, two samples might reasonably be discarded because of probable contamination as already explained. On the same basis the average iron content at different depths in five holes that are about 30 ft. each in depth is: upper 10 ft., 52.8 per cent.; from 10 to 20 ft. depth, 53.4 per cent.; and from 20 to 30 ft., 48.2 per cent., or 52.2 per cent. if one of the poor samples previously mentioned be excluded. These figures indicate that there is a slight increase in the iron content below the surface, followed by a decrease near the underlying rock. A similar variation is shown in holes of greater depth. A like increase in the content of iron in the ore immediately below the surface is a feature of the Cuban deposits, also, according to the papers previously quoted.

The only detailed analysis of the Surigao ore was performed upon a sample taken from the surface by Mr. Cameron. This analysis follows:[12]

	Per Cent.
Hygroscopic water	13.50
Combined water	6.60
Silica	1.04
Alumina	10.56
Ferric oxide	66.80
Ferrous oxide	0.36
Chromium oxide	1.15
Nickel oxide	nil
Sulphur	trace
Phosphorus	trace
Total	100.01
Metallic iron, dried ore	54.29
Metallic iron, ore deprived of combined water	58.20

In its lack of nickel, as represented by the single analysis, the Surigao ore is different in an important respect from the Cuban ore.

Quantity of Lateritic Iron Ore

In attempting to ascertain what quantity of ore is present in the Surigao deposit and what percentage of iron is contained in the average ore, hand drills were employed to determine the thickness of the ore mantle at different places and to secure representative samples of the ore from different depths. The area covered by iron ore is 40 square miles; although it was decided after inspection that about 12 square miles of this area is too distant from the coast and too mountainous to be considered economically important at present, the remaining area is much too large to admit of close sampling in a preliminary study. Con-

[12] Analysis by Francisco Peña, Bureau of Science.

sequently, groups of drill holes were placed so as to sample thoroughly small tracts in different parts of the economically important area. In all, 98 drill holes were sunk, admittedly too small a number of tests to establish closely and beyond question the composition and thickness of the ore mantle over an area of 28 square miles; nevertheless, the results obtained are so uniform even in widely separated parts of the deposit that they are believed to indicate, at least roughly, the tonnage and quality of the ore.

Of the 28 square miles of economically important area, only 12 square miles proved to be covered with ore of important thickness. The remaining portions of the deposit are either partly denuded because of the steepness of the slopes or are made up of alluvial-filled valleys in which the ore is contaminated with sand and gravel derived from the parent rock. Drill holes were placed so as to test four widely separated sections of the areas over which the ore mantle is relatively thick and one section in which the ore is thin and discontinuous, the holes in all cases being located at regular intervals, generally at the corners of 1,000-ft. squares. Of the holes located in areas over which the ore is of good depth, 6.8 per cent. fell upon bare places and encountered no ore; 28.8 per cent. encountered from 2 to 10 ft. of ore; 34.3 per cent. encountered from 10 to 20 ft. of ore; 21.9 per cent. encountered from 20 to 30 ft. of ore; 6.8 per cent. encountered from 30 to 40 ft. of ore; and 1.4 per cent. encountered from 40 to 50 ft. of ore. Of the holes in the poor area, 29.4 per cent. encountered no ore; 58.8 per cent. encountered from 2 to 10 ft. of ore; 5.9 per cent. encountered from 10 to 20 ft. of ore; and 5.9 per cent. encountered from 20 to 30 ft. of ore.

From these figures and the weight of a unit volume of the ore in place it can be calculated that the total economically important metric tonnage is 430,000,000, of which about 375 000,000 tons is contained in that part of the ore mantle which is 10 ft. or more in thickness. Reasonably accessible from the coast, but divided into a number of separate deposits by precipitous valleys, there is 275,000,000 tons of ore, with 260,000,000 tons lying 10 ft. or more deep. Finally, within two areas of ore which could be mined from a base at Dahikan Bay, the most feasible harbor site, there is 138,000,000 tons, 130,000,000 tons of which is more than 10 ft. thick. It should be noted, however, that the bulk of even this most favorably situated ore lies on the tops of hills and broad divides from 400 to 1,000 ft. above sea level and that within each of the two areas near Dahikan Bay there are ravines and denuded slopes which would of necessity be avoided in mining.

The Formation and Distribution of Residual Iron Ores

BY C. L. DAKE,* M. A., ROLLA, MO.

(San Francisco Meeting, September, 1915)

Residual deposits occur both as products of weathering and as products of hydrothermal decay.

Products of Weathering

That climatic conditions affect greatly both the rate and the results of weathering, is shown in temperate, polar, arid, and tropical regions, each of which will be considered separately.

Weathering under Temperate Climates

The decomposition of any rock is accomplished by the removal of the more soluble bases, and a concentration of the relatively insoluble oxides of iron, aluminum, and silicon. By reason of its somewhat greater solubility, silica is much more subject to removal than iron and alumina. The two important residual end products of weathering are sands and clays, the former consisting essentially of quartz, and the latter of kaolin and limonite.[1]

When such deposits are formed *in situ*, the sands and clays are not separated, but remain as mixtures, which can be more or less perfectly separated by treatment with water. In such a separation, the ferric oxide or hydrate tends, by reason of its fine state of division, to remain with the clays and silts rather than the sands. While ferruginous sandstones are everywhere common, they are rarely rich enough in iron to become an ore without secondary concentration. Since the silica of these sandstones is completely crystalline quartz, it is much less liable to removal by solution than are the more soluble forms of partly amorphous chert and jasper, which have been removed on so large a scale from the Lake Superior iron formations.[2] Moreover, the iron cement of sandstones, which constitutes the chief iron content of these rocks, is often a result of later infiltration, and not of original sedimentation. Thus if clays rather than sands contain the iron of residual decay,

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[1] Merrill, G. P.: *Rocks, Rock-Weathering, and Soils*, 2d ed., p. 289 (1906).

[2] Van Hise, C. R., and Leith, C. K.: Geology of the Lake Superior Region, *Monograph lii, U. S. Geological Survey*, p. 537 (1911).

we are led to consider the former more carefully with a view to determining, as far as possible, the behavior of the iron with respect to the other constituents.

Since sands and clays, together with their cemented equivalents, sandstones, quartzites, shales, and slates, already represent the ultimate products of weathering, and consist of the more insoluble rock elements, we do not look to them to furnish residual products by further decay. In exceptional cases, such decay seems to go on. D'Invilliers[3] accounts for certain iron ores in Pennsylvania by the decay of damourite slates; but in general, sandstones and slates are the ultimate products beyond which chemical decay, under normal weathering conditions, cannot go.

Limestones, on the other hand, while representing an ultimate product of decay, are easily soluble, and if they contain insoluble impurities, these will be concentrated by solution decay. Such concentration is widely observed everywhere. Where limestone contains iron as one of the important impurities, the residuary products will be rich in iron.

The processes of decay already described are also active on igneous rocks, and result in residual clays and sands. Both acid and basic rocks are attacked; but in general, the more basic the rock, the more rapid the decay, and the greater the percentage of iron in the resulting clay. This results from two causes. First, the SiO_2 in the more basic rocks is all combined, and when freed by the removal of the bases, is in the colloidal form, and more soluble than the quartz of an acidic rock; and for that reason, its removal in solution will result in a higher degree of concentration. Secondly, basic rocks are, as a rule, originally higher in iron than acidic rocks. Everywhere in temperate regions where the humidity is high, such decay goes on rapidly and to great depths. The region of the southern Appalachians is a good example, with its deep soils and their brilliant reds and yellows.[4]

Concerning the efficiency of weathering in increasing iron content, considerable data are available. Chamberlin[5] gives four limestone-clays from the Driftless Area of Wisconsin, containing respectively 5.52, 8.53, 10.04, and 17.19 per cent. of ferric oxide.[6] Analyses of the parent limestone are not given, but Grant[7] gives two analyses of typical lime-

[3] D'Invilliers, E. V.: Iron Ore Mines of the Great Valley, *Annual Report, Pennsylvania Geological Survey*, 1886, pt. 4, p. 1411.

[4] Russell, I. C.: Subaerial Decay of Rocks and Origin of the Red Color of Certain Formations, *Bulletin No.* 52, *U. S. Geological Survey*, p. 22 (1889).

[5] Chamberlin, T. C., and Salisbury, R. D.: Preliminary Paper on the Driftless Area of the Upper Mississippi Valley, *Sixth Annual Report, U. S. Geological Survey*, p. 250 (1884–85).

[6] For the sake of uniformity, the FeO is recomputed as Fe_2O_3 in this and all the following comparisons.

[7] Grant, U. S.: Lead and Zinc Deposits of Wisconsin, *Bulletin No.* 14, *Wisconsin Geological and Natural History Survey*, pp. 13, 33 (1906).

stones from the same area and formation, showing respectively 0.9 and 0.95 per cent. of Fe_2O_3. The Karst limestone, with only 0.044 per cent. of ferruginous silicates, yields the famous Terra Rossa, with a marked iron content.[8] Other cases of limestone weathering show concentration of iron as follows:

		Limestone Per Cent. Fe_2O_3	Residual Soil Per Cent. Fe_2O_3
Virginia[9]		0.19	10.5
Knox dolomite[10]		0.23	8.3
Lexington, Va.[10]	$\left.\begin{matrix}Fe_2O_3\\Al_2O_3\end{matrix}\right\}$	0.42	15.16
Bermuda (red earth soils)[10]	$\left.\begin{matrix}Fe_2O_3\\Al_2O_2\end{matrix}\right\}$	0.54	13.898

Weathering of acid igneous rocks shows concentration of iron as tabulated below:

	Unweathered Rock Per Cent. Fe_2O_3	Residual Soil Per Cent. Fe_2O_3
Micaceous gneiss[11]	9.06	12.18
Granite[12]	4.00	4.39 (and FeO)
Biotite granite[13]	1.45	2.44
Biotite granite[14]	1.41	1.91
Biotite granite[15]	1.96	6.33
Porphyritic biotite granite[16]	2.51	3.45
Granite[17]	3.46	5.04
Granite[18]	1.44	2.05
Hornblende granite[18]	3.43	2.67

Acid rocks in general show upon decay a concentration of iron which is slight, as compared with that shown by limestones.

Below are tabulated examples of the concentration of iron shown by the weathering of basic igneous rocks.

[8] Merrill, G. P.: *Rocks, Rock-Weathering, and Soils*, 2d ed., p. 375 (1906).

[9] Watson, T. L.: Lead and Zinc Deposits of Virginia, *Bulletin No.* 1, *Virginia Geological Survey*, p. 98 (1905).

[10] Russell, I. C.: *Bulletin No.* 52, *U. S. Geological Survey*, p. 24 (1889).

[11] Merrill, G. P.: Weathering of Micaceous Gneiss in Albermarle County, Virginia, *Bulletin of the Geological Society of America*, vol. viii, p. 157 (1896).

[12] Merrill, G. P.: Disintegration of the Granitic Rocks of the District of Columbia, *Bulletin of the Geological Society of America*, vol. vi, p. 321 (1894).

[13] Watson, T. L.: Granites and Gneisses of Georgia, *Bulletin No.* 9A, *Georgia Geological Survey*, p. 302 (1902).

[14] *Idem*, p. 312.

[15] *Idem*, p. 315.

[16] *Idem*, p. 321.

[17] Holland, P., and Dickenson, E.: *Proceedings of the Liverpool Geological Society*, vol. vii, pt. 1, p. 116.

[18] Culver, H. E.: The Formation of Laterites: A Thesis Submitted for the Degree of Master of Philosophy, at the University of Wisconsin, p. 39 (1911).

	Fresh Rock Per Cent. Fe_2O_3	Residual Soil Per Cent. Fe_2O_3
Diorite[19]	....	15.94
Greenstone[20]	....	18.00
Diorite[21]	....	18.00
Diabase[22]	11.54	13.37
Diabase[23]	11.60	35.96
Diabase[24]	13.54	40.68

Residual decay in temperate regions yields sediments rich in iron as compared with the original rocks. Only limestones and basic igneous rocks, however, show sufficient concentration to produce deposits rich enough to approach ores.

Weathering in Polar Regions

Polar regions with their low temperatures are regions of disintegration rather than of decomposition.[25] Such chemical decay as has occurred seems to be of the same general type as that of the temperate regions, the presence of ocherous clays[26] indicating that some chemical decomposition goes on. Such decay, however, as far as observed, is not sufficient to produce any large bodies of iron-rich sediments.

Weathering in Arid Regions

In arid regions disintegration is also much more characteristic than decomposition.[27] In the valley of Lower California, sediments are forming which consist largely of undecomposed rock material. Walther concludes[28] that chemical weathering in the desert is of slight importance. Even where such weathering does go on in the desert, the soluble constituents tend to remain, because the water supply is too slight to remove them.[29] If this is true, there would be little concentration of

[19] Ries, H.: Clays of Wisconsin, *Bulletin No.* 15, *Wisconsin Geological and Natural History Survey*, p. 117 (1906).

[20] *Idem*, p. 131.

[21] *Idem*, p. 174.

[22] Holland and Dickenson: *Proceedings of the Liverpool Geological Society*, vol. vii, p. 108.

[23] Watson, T. L.: Weathering of Diabase near Chatham, Virginia, *American Geologist*, vol. xxii, No. 2, p. 87 (Aug., 1898).

[24] Merrill, G. P.: Disintegration and Decomposition of Diabase at Medford, Massachusetts, *Bulletin of the Geological Society of America*, vol. vii, p. 350 (1895).

[25] Van Hise, C. R.: A Treatise on Metamorphism, *Monograph XLVII, U. S. Geological Survey*, p. 498 (1904).

[26] Walther, J.: *Einleitung in die Geologie*, vol. iii, p. 740.

[27] Van Hise, C. R.: *Op. cit.*, p. 496.

[28] Walther, J.: *Op. cit.*, p. 777.

[29] Van Hise, C. R.: *Loc. cit.*

insoluble material, since this concentration is normally brought about by the removal of the soluble constituents.

Hilgard,[30] on the other hand, shows a distinctly higher percentage of iron in desert than in humid soils. He states, however, that there is no obvious chemical reason for this greater abundance, and assigns as a possible cause that the soils chosen to represent arid regions resulted in many cases from rocks originally high in iron.

Residual deposits of iron are probably not forming in desert regions in quantities worthy of consideration.

Tropical Weathering

In tropical regions, weathering is more intense than in temperate climates, and proceeds more rapidly and to greater depths. There is also, in certain regions, a difference in the residual products; instead of the characteristic clays of temperate regions, laterites often result. It seems to be well established, however, that true clays as well as laterites are formed in tropical regions.

Lateritic Deposits.—Since the term laterite has been so loosely used, it is necessary to adopt a definition as a basis for further discussion. Clarke[31] defines laterite as "essentially a mixture of ferric hydroxide, aluminum hydroxide, and free silica in varying proportions." Culver,[32] who has made a close study of laterites, follows Clarke in his definition, and the term will be thus limited in this discussion. As defined above, laterites differ from true clays in the absence of the aluminum silicate, kaolin, and the presence in its place of some aluminum hydrate. Culver, in the work cited above, concludes, from the examination of all the available data, that laterites develop only in tropical and semi-tropical regions of heavy rainfall and only from basic rocks. The wide extent and great depth of lateritic deposits make their consideration of prime importance in the study of iron-rich residual deposits. Moreover, laterites represent a more complete decay of rocks and a greater removal of silica than do ordinary clays. Therefore they must show a proportionate increase in the remaining constituents, and consequently, other factors remaining constant, a residual deposit richer in iron. To illustrate, Culver[33] gives an example of a clay which contains 47 per cent. SiO_2, and 14.6 Fe_2O_3. He also gives a list[34] of typical laterites showing the content of SiO_2 from 0.37 to 10.75, with an average of less than 3

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[30] Hilgard, E. W.: *Soils*, p. 392.

[31] Clarke, F. W.: The Data of Geochemistry, *Bulletin No.* 491, *U. S. Geological Survey*, p. 470 (1911).

[32] Culver, H. E.: The Formation of Laterites: A Thesis Submitted for the Degree of Master of Philosophy at the University of Wisconsin (1911).

[33] *Idem*, p. 53.

per cent. If the SiO_2 were removed from the above clay to form a typical laterite, it would mean a loss of SiO_2 expressed by the following equation: $\frac{100 - x}{100} \times 3 = 47 - x$,[35] from whence $x = 45.36$ per cent. of the original weight. The residual portion would then be represented by $100 - 45.36 = 54.65$ per cent. of the original weight. The iron content of the residual deposit would then be increased from 14.6 per cent. of the original weight to $\frac{100}{54.64}$ of 14.6, or 26.72 per cent. of the weight of the residual mass. This indicates clearly one reason for the high content of iron in laterites. Another reason is doubtless their formation from basic rocks only. The degree of concentration of iron shown in lateritic decay is expressed in the following table:

	Fresh Rock Per Cent. Fe_2O_3	Residual Soil Per Cent. Fe_2O_3
Epidiorite (British Guiana)[36]	4.66	25.60 26.73
Diabase (British Guiana)[37]	10.70	22.31 24.82
Dolerite (India)[38]	13.69	23.40

A series of laterites from India[39] run 13.75, 23.41, 26.61, 34.37, 37.88, 47.27, 51.25, and 56.01 per cent. Fe_2O_3. The parent rock is not given. Two laterites from Surinam[40] contain 55.94, and 62.08 per cent. ferric oxide. Merrill[41] cites an Indian laterite with 55.5 per cent. Fe_2O_3. The parent rock is not given. Culver[42] cites certain unpublished data furnished by Prof. W. J. Mead, of the University of Wisconsin, on the lateritic iron ores of Cuba. The original is a serpentine rock, the freshest of which contains 9.8 per cent. Fe_2O_3. The residual lateritic ore, sampled every foot from surface to bed rock, 28 samples in all, averages

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[34] Culver, H. E.: *Idem*, pp. 56 to 60.

[35] Let x = loss of SiO_2 in per cent. of original rock.
Then $47 - x$ = remaining SiO_2 in per cent. of original rock.
And $100 - x$ = total remaining material in per cent. of original rock.
But 3 = remaining SiO_2 in per cent. of total remainder.
And $\frac{100 - x}{100} \times 3$ = remaining SiO_2 in per cent. of original rock.
Whence $\frac{100 - x}{100} \times 3 = 47 - x$ and $x = 45.36$ per cent.

[36] Culver, H. E.: *Op. cit.*, p. 30.

[37] *Idem*, p. 43.

[38] *Idem*, p. 54.

[39] *Idem*, pp. 59 to 60.

[40] *Idem*, p. 62.

[41] Merrill, G. P.: *Rocks, Rock-Weathering and Soils*, 2d ed., p. 299 (1906).

[42] Culver, H. E.: *Op. cit.*, p. 45.

64.3 per cent. Fe_2O_3. The data above show that lateritic decomposition is very efficient in the concentration of iron, in residual deposits.

Tropical Clays.—Culver, in the paper cited above, says, "No deposit of true laterite is known which has certainly been derived from acid igneous rocks." He quotes Harrison's extensive work on laterites as showing conclusively that, under conditions where true laterites are formed from basic rocks, acidic rocks alter to true clays, with the normal kaolin content—clays entirely comparable to those of temperate regions.

General Distribution of Weathering Products

It may be said in general that residual deposits are not formed in desert and frigid regions. Such deposits, on the other hand, occur in abundance in humid temperate and tropical regions. Only areas covered either by limestones or basic igneous rocks yield residual deposits high in iron. True laterites, which represent the residuary material richest in iron, are developed only over areas of basic rocks, in tropical regions of heavy rainfall.

PRODUCTS OF HYDROTHERMAL DECAY

Hydrothermal metamorphism is frequently so profound as to change materially the chemical and mineralogical composition of large areas of rocks. It is only rarely, however, that this process can actually be seen occurring at the present time, under conditions which permit the changes to be traced through their various steps. It is still more exceptional to find recognizable cases where this alteration has gone to so complete a stage that the end products consist of residual soils, essentially in the ultimate stage of decay. So far as the writer knows, Maxwell[43] is the only investigator who has made a careful study of this method of formation of residual soils.

Nature of Hydrothermal Decay in the Formation of Soil

The rocks with which Maxwell deals, in the investigation above referred to, are wholly of basaltic type. He divides them into non-hydrous lavas, hydrous lavas, and tufas. These divisions are based upon what he assumes to be the original or primary alteration effected upon the lavas by steam and heated waters at the time of their extrusion. Originally representing the same magmas, they now differ in the degree of hydration and oxidation they have undergone. The iron content of these lavas is as follows:

[43] Maxwell, Walter: Lavas and Soils of the Hawaiian Islands: Rept. of the Experimental Station of the Hawaiian Sugar Planters' Association, *Special Bulletin A, Department of Agriculture and Chemistry, Honolulu* (1905).

	FeO Per Cent.	Fe_2O_3 Per Cent.	Total Iron Recomputed as Fe_2O_3 Per Cent.
Non-hydrous lavas.....	8.43	4.04	13.41
Hydrous lavas.........	8.66	5.62	15.24
Tufas (Tuffs)..........	1.69	25.79	27.67

The above are the averages of a large number of individual analyses of each type. Two important points are brought out in the above figures: First, there is a definite relative increase of the ferric over the ferrous oxide in the change from non-hydrous lavas to tuffs, and second, there is an apparent increase in the total iron content. This apparent increase is probably due to the loss of the soluble constituents, the silica decreasing from 49.12 to 26.60 and the lime from 9.24 to 1.41 per cent. It will at once be noted that the change is quite comparable to the lateritic process just discussed.

After the lavas have been ejected and solidified, there are large areas in the Hawaiian Islands where solfataric action still further modifies the composition of the rocks. These changes are brought about by the action of intensely acid steam, in which the sulphuric acid sometimes reaches 5 per cent. or more. From the floors of these solfataras, blocks of partly altered lava were taken, while yet hot and while being acted upon by the steam ascending through the fissures in the floor. These showed the lava in all stages of decay. A peculiarity of the outcome of the process is that all the products seem to be in a sense residual: gypsum, silica, alum, ocher. The products are, moreover, segregated in alternating layers, irregular nodules, vast pockets, and other complex relations. Large masses of white siliceous soils alternate with areas of red ocherous earths, and these in turn with gypsum. The essential point of his discussion, however, as applied to the theme of this paper, is that hot waters and gases, carrying abundant sulphuric acid, have been proved to effect the complete decomposition of basic lavas, and the segregation of the products of the decay, so that certain of the resultant soils show a very marked increase in the proportion of ferric oxide, as compared with that in the original rocks.

Efficiency of Hydrothermal Decay in Increasing Iron Content

It must be borne in mind, in discussing the increase in iron due to decay by hydrothermal agencies, that the residual deposits vary widely in character. The following is a list of the more common end products, with their iron content, all of which are said to exist in large quantities.

	Fe_2O_3 Per Cent.
Gypsum soils....................................	0.7
Alum soils......................................	12.3
Impure siliceous residue........................	4.9
Ocherous deposits...............................	44.5

In discussing those residual soils of Hawaii, which he believes to be the result of the action of steam and hot water, Maxwell classes them according to color. Eighteen analyses of white and gray soils show from 0.7 to 18.62 per cent. Fe_2O_3, with an average of 6.01 per cent. The Fe_2O_3 content of nine brown and yellow soils varies from 3.25 to 26.68 per cent., and averages 10.82 per cent. Twenty analyses of red soils vary between 16.99 and 83.68 per cent. in Fe_2O_3 and average 41.37 per cent. These figures show beyond a doubt that the red soils, which are so abundant in the islands, are rich in iron. These soils among others are ascribed by Maxwell to the action of hot water and steam. In summing up, he gives the following figures, compiled from a large number of individual cases, as the iron content of the average lava and the average soil of the islands: lavas, 13.36 per cent.; and soils, 36.45 per cent., of FeO and Fe_2O_3.

Over what areas hydrothermal alteration is now, or has been in the past, effective in the enrichment of iron sediments is a question not easy to answer. It seems entirely probable that in regions of intense effusive activity, especially where the basic and more easily decayed types of lava are found, conditions might prevail entirely similar to those found by Maxwell in the Hawaiian Islands. Further investigation along this line would be very desirable.

It is quite possible that the decay of the Deccan traps[44] and the basalts of Antrim,[45] resulting in their accompanying iron-rich deposits, as well as other cases of a similar nature, may have been due in part, at least, to thermal alteration at or shortly after extrusion. The scale on which this alteration has taken place in the Hawaiian Islands leads us to believe that hydrothermal decay might be an important contributory factor in the formation of iron-rich residual deposits.

Alteration in Situ of Previously Formed Iron Deposits

The oxidation of cherty iron carbonate, and the subsequent leaching of the silica, with resultant enrichment of the iron formation to form ore,[46] as well as the weathering of pyrites to form gossan orebodies,[47] is a type of weathering resulting in what may be called residual deposits. But since this paper deals only with the primary accumulation of iron formations, such local secondary enrichment may be dismissed without further mention here.

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[44] Foote: *Memoirs of the Geological Survey of India*, vol. xii, p. 200.

[45] Geikie, Sir A.: *Ancient Volcanoes of Great Britain*, vol. ii, p. 204.

[46] Van Hise, C. R., and Leith, C. K.: The Geology of the Lake Superior Region, *Monograph LII, U. S. Geological Survey*, p. 529 (1911).

[47] Moxham, E. C.: The Great Gossan Lead of Virginia, *Trans.*, xxi, 133 (1892–93).

Summary

Residual soils rich in iron are produced both by weathering and by hydrothermal alteration.

Weathering produces important accumulations of iron sediments only from limestones and basic igneous rocks. In other rocks, such concentration is of minor importance.

Only temperate and tropical regions of moderate to heavy rainfall yield residual weathering products in abundance. Polar and arid regions are more favorable to mechanical disintegration.

Laterites, which by reason of their lower silica content and the nature of the parent rock are richer in iron than ordinary residual clays, are developed only from basic igneous rocks, and in tropical regions of heavy rainfall. Such laterites are widespread in the tropics, and represent enormous concentration of residual iron.

Hydrothermal decay has, in the Hawaiian Islands, produced iron-rich residual soils very comparable to laterites. Observation is limited, and we know little of the distribution or importance of this type of residual deposits.

Of the above types, hydrothermal decay, normal weathering, and lateritic weathering, the latter is probably of the greatest importance. This is due both to the great abundance and to the higher iron content of the laterites.

The Formation and Distribution of Bog Iron-Ore Deposits

BY. C. L. DAKE,* M. A., ROLLA, MO.

(San Francisco Meeting, September, 1915)

Method of Formation of Bog Ores

Chemistry of Iron Solution

Iron is much more soluble in the ferrous than in the ferric form. Where, as in the case of the ferrous silicates and the sulphides, the iron is already in the ferrous form, it may go at once into solution and be removed, upon the breaking down of the parent rock. If in the ferric form, it must first be reduced; usually by the decay of organic matter, the demand of the decaying carbonaceous matter for oxygen being so great that the latter is abstracted from the ferric compounds and the iron is thereby reduced and rendered soluble. The iron may go into solution as the sulphate, or in the presence of excess CO_2 as ferrous carbonate. It is probably soluble as salts of the organic acids.[1] Recent work indicates that these are important in the formation of bog ores.[2] There is considerable variety of opinion concerning the behavior of these organic compounds.[3]

The secretions about the roots of plants and trees also cause the solution and removal of iron and a consequent bleaching of the soil.[4]

While the matter is not yet well understood, we can probably say definitely that the so-called organic acids are a group of colloids which possess slightly acid properties, and the salts of which are partly soluble and partly insoluble.[5] Their iron salts readily change from the ferrous to the ferric form, and their solubility is greatly modified by the presence of ammonia.

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[1] Beck, Richard: *The Nature of Ore Deposits*, vol. i, pp. 101, 102 (1905.)

Julien, A. A.: Geological Action of the Humus Acids, *Proceedings of the American Association for the Advancement of Science*, vol. xxviii, p. 311 (1879).

[2] Aschan, Ossian: Die Bedeutung der wasserlöslichen Humusstoffe für die Bildung der See-und Sumpferze, *Zeitschrift für praktische Geologie*, vol. xv, No. 2, p. 56 (Feb., 1907).

[3] Aschan, Ossian: *Loc. cit.*

Julien, A. A.: *Loc. cit.*

Hilgard, E. W.: *Soils*, pp. 122, 123 (1906).

[4] Phillips, J. A., and Louis, H.: *A Treatise on Ore Deposits*, p. 36 (1896).

[5] Clarke, F. W.: The Data of Geochemistry, *Bulletin No.* 491, *U. S. Geological Survey*, p. 98 (1911).

Chemistry of Iron Precipitation

The simplest cause of precipitation of iron is oxidation from the soluble ferrous to the much less soluble ferric compounds. Ferrous carbonate, soluble in carbonated waters, is also precipitated as the carbonate by evaporation of the excess CO_2. Certain bacteria precipitate iron by building the hydrated oxide into their body cells.[6] Iron is precipitated as the sulphide when vegetation decays in the presence of sulphates of lime and magnesia, with a deficiency of oxygen.[7] There are many other means[8] by which, under special conditions, iron may be precipitated in nature. Oxidation, evaporation of CO_2, and iron bacteria have usually been held to be most important.

Character of Bog Ores

Bogs are often accompanied by thin layers of iron ore, consisting largely of hydrated ferric oxide, frequently with considerable amounts of ferrous carbonate, and sometimes ferrous sulphate and silicate.[9] In marine marshes, under the influence of the sulphates of sea water, iron may go down as the sulphide.[10] Bog ores, however, consist essentially of limonite, with subordinate amounts of carbonate. Such ores are usually high in phosphorus[11] and almost invariably preserve abundant fossil fragments of leaves and plant remains, which serve to identify them as of bog origin.[11]

Distribution of Bog Ores

Relations to Geographical Latitude

Those countries best known as producers of bog ores are Norway, Sweden, Germany, Poland, Finland, and Canada.[12] Many deposits

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[6] Winchell, N. H. and H. V.: The Iron Ores of Minnesota, *Bulletin No.* 6, *Minnesota Geological and Natural History Survey*, p. 223.
Beck, Richard: *The Nature of Ore Deposits*, vol. i, pp. 101, 102 (1905).
Bischof, Gustav: *Elements of Chemical and Physical Geology*, vol. i, p. 169 (1854).

[7] Dawson, Sir William: *Geology of Nova Scotia*, p. 24.
Geikie, Sir A.: *Text-Book of Geology*, 4th ed., p. 612 (1903).
Bischof, Gustav: *Op. cit.*, p. 162.

[8] Kemp, J. F.: *Ore Deposits of the United States and Canada*, 5th ed., p. 87 (1903).
Beck, Richard: *Loc. cit.*

[9] Moore, E. J.: Occurrence and Origin of Some Bog Iron Deposits in the District of Thunder Bay, Ontario, *Economic Geology*, vol. v, No. 6, p. 528 (Sept., 1910).

[10] Dawson, Sir William: *Loc. cit.*

[11] Winchell, N. H. and H. V.: *Op. cit.*, p. 221.
Phillips, J. A., and Louis, H.: *Op. cit.*, p. 42.
Kemp, J. F.: *Op. cit.*, p. 89.

[12] Roth, Justus: *Allgemeine und chemische Geologie*, vol. i, pp. 597, 598, 645.
Walther, J.: *Einleitung in die Geologie*, p. 751.
Beck, Richard: *Op. cit.*, p. 100.
Bischof, Gustav: *Loc. cit.*
Phillips and Louis: *Op. cit.*, p. 37.

occur in the northern United States.[13] A few localities are known in regions farther south, among them the deposits of Missouri,[14] North Carolina,[15] and Rio Tinto, Spain.[16] Exhaustive lists of bog-ore districts, however, include few localities outside the limits of the regions just named. Careful examination has shown that the overwhelmingly larger number of the known bog deposits occur in the northern portions of the three northern continents.

Relations to Glaciation

A very small percentage of our bog-ore deposits is to be found outside the glaciated regions, while within those regions they are characteristic and abundant, both in Europe and in North America.[17] The ores accumulate in the smaller swamps, while in large neighboring morasses they may be entirely wanting, probably by reason of the fact that the concentration is greater in the smaller swamps.[18] Glaciated areas are more favorable to iron accumulation, because they present numerous small swamps, rather than the immense ones, such as are common to the lower flood plains and deltas of large rivers, or the saline marshes of low coastal plains.

Relation to Sources of Supply

Another factor, doubtless, is the proximity of available sources of supply for the iron. From 1 to 3 per cent. of the glacial drift is said to be made up of magnetite;[19] and other iron compounds are present. The unconsolidated condition of the drift renders this iron content easily accessible. The glaciated regions parallel somewhat closely the so-called "Pre-Cambrian Shields" of both Europe and North America, areas in which enormous amounts of older iron-rich rocks are localized. This may also account in part for the noteworthy supply of iron for bog deposits in both drift and bed rock of the glaciated regions.

Of the localities of bog ores cited outside the glaciated area, three at least are definitely related to local sources of iron. In the case of the Rio Tinto beds the conditions are as follows:[20] A marsh or shallow lake ex-

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[13] Kemp, J. F.: *Op. cit.*, pp. 89, 90.

[14] Nason, F. L.: Iron Ores of Missouri, *Missouri Geological Survey*, vol. ii, p. 182 (1892).

[15] Nitze, H. B. C.: Iron Ores of North Carolina, *Bulletin No.* 1, *North Carolina Geological Survey* (1893).

[16] Phillips and Louis: *Op. cit.*, pp. 40 to 42.

[17] Shaler, N. S.: General Account of the Fresh-Water Morasses of the United States, *Tenth Annual Report, U. S. Geological Survey*, pt. 1, p. 305 (1888–89).

[18] *Op. cit.*, p. 306.

[19] *Op. cit.*, p. 305.

[20] Phillips and Louis: *Op. cit.*, pp. 40 to 42.

tended over the area where the bog ores are now found, and into this flowed iron solutions resulting from the decomposition of extensive beds of pyrites. From these chalybeate waters, the bog ores were, and are still, thrown down. In Park County, Colo.,[21] water which percolates through the rock is charged with iron sulphate, resulting from the solution of beds of pyrites in the vicinity. The iron is thrown down in a small marsh, through which the drainage of the region passes. In the Missouri deposits, a very ferruginous sandstone is underlain by an impervious shale. Bogs form along the contact where the percolating waters escape, and here the ores are deposited.[22] From the above three cases, it will be seen that an immediate source of iron may become one of the factors in localizing such deposits.

Relation to Climate and Its Chemical Effect

The prevailing red color of soils in the tropics has been assumed to be a result of a higher oxidation of the iron and its superficial dehydration. It seems possible that the rapid decay of vegetation in tropical regions, and the resulting lack of accumulation of partly decayed vegetable matter, may effect the accumulation of bog iron ores.

Two alternative suggestions have been tentatively advanced: first, that iron does not go into solution in the tropics as abundantly as in colder regions; and secondly, that it goes into solution abundantly, but is not as readily precipitated.

Iron Solvents in Tropical Soils.—Earth with a high content of partly decayed vegetation is found to relatively slighter extent in the tropics than elsewhere, because the conditions which favor rapid growth favor rapid decay also. The lack of large quantities of humus in the tropical soils, except at high altitudes,[23] is pretty generally reported. Peat accumulates only where there is a nearly permanent water supply, and areas of accumulation are few outside the glaciated region, by reason of high temperature, long summers, frequent drought, and lack of undrained areas.[24] It must be borne in mind, however, that the accumulation of partly decayed woody matter is not necessarily indispensable to the development of abundant organic acids. In this connection Hil-

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[21] *Loc. cit.*

[22] Nason, F. L.: *Op. cit.*, p. 182.

[23] Taylor, A. E.: The Peat Deposits of Indiana, *Thirty-first Annual Report, Department of Geology and Natural Resources, Indiana*, p. 75 (1906).

Walther, J.: *Einleitung in die Geologie*, pp. 745, 752, 811.

Roth, Justus: *Allgemeine und Chemische Geologie*, vol. ii, p. 645.

Schimper, A. F. W.: *Plant Geography on a Physiological Basis*, p. 380 (1903),

[24] Davis, C. A.: Preliminary Report on the Peat Deposits of North Carolina *Economic Paper No.* 15, *North Carolina Geological and Economic Survey*, pp. 147, 148 (1908).

gard,[25] somewhat in opposition to the general conception, concludes that tropical soils are likely to be rich in humus, where the interval between the rainy seasons is not too long; otherwise, an almost complete oxidation of the humus must take place, leaving only a trifling organic residue. Organic acids are drained away to such an extent that during the rainy season many tropical rivers become deep coffee brown.[26]

The evidence seems to be somewhat contradictory. Where tropical soils are subjected to long and frequent drought, humus is rare. In tropical regions of nearly continuous heavy rainfall, humus is said to be abundant. Excessive decay must yield plentiful CO_2. Growing roots must yield abundant solvents. Tropical soils, then, may or may not carry iron solvents, the distribution depending largely on drought and rainfall.

Iron Solvents in Tropical Rivers.—Many tributaries of the Amazon and Orinoco are colored nearly black by organic acids.[27] The following tabulation gives for several well-known rivers the amount of organic matter:[28]

River	Organic Matter, Per Cent.	Salinity, Parts per Million
Danube	3.25	184
James	4.14	89
Nile	10.36	119
Hudson	11.42	108
Cumberland	12.08	124
Amazon	15.03	59
Delaware	16.00	70
Xingu	20.63	45
Tapajos	24.16	38
Plata	49.59	91 to 206
Negro	53.89	132
Uruguay	59.90	66

There is evidently a decided tendency in tropical rivers to carry a higher percentage of organic matter than do those of temperate zones. Their total salinity, however, is rather lower. We have no data as to the relative amounts of dissolved CO_2 carried by tropical and temperate rivers, but it is believed that the abundant decaying vegetation in tropical waters would yield much CO_2. We may then safely say that solvents for iron are not lacking in tropical streams.

Iron Content of Tropical Rivers.—In only a few analyses of tropical waters have separate determinations of Fe_2O_3 and Al_2O_3 been made.

[25] Hilgard, E. W.: *Soils*, pp. 129, 399 (1906).

[26] Schimper, A. F. W.: *Loc. cit.*

[27] Clarke, F. W.: The Data of Geochemistry, *Bulletin No.* 491, *U. S. Geological Survey*, p. 82 (1911).

[28] *Op. cit.*, p. 97.

Analyses of Tropical River Waters.[29]

River	Fe_2O_3 Parts per Million	Total Salinity Parts per Million
Plata	4.0131	91
Paràna	3.1458	98
Nile	3.1535	119

Analyses of Temperate River Waters for Comparison[29]

Mississippi at New Orleans	0.2158	166
St. Lawrence at Ogdensburg	0.0804	134
Danube near Vienna	0.3340	167

The three tropical rivers in the above list are trunk streams, from which the water was taken, well down toward the mouth. As will be noted, they run far higher in iron than the three streams in temperate regions cited for comparison. The analyses given are too few in number to warrant the positive statement that tropical rivers in general carry a high iron content; but the figures are suggestive.

The Precipitation of Iron in Tropical Regions.—It has already been shown that bacteria aid in the precipitation of iron. Whether tropical conditions increase or decrease the activity of such bacteria is unknown.

It is possible that the high content of the rivers in solvents may tend to prevent precipitation. I know of no careful determination of organic matter or dissolved CO_2 in the waters of those bogs in which iron is now going down. In Scandinavian lakes where lake ores are forming, there is a marked content of humic acids;[30] and Lough Neagh, in Ireland, which has a high organic content,[31] carries also 10.416 parts per million of Fe_2O_3.[32] In the latter case, the water seems to be high both in iron and in organic matter; but I have seen no positive statement that bog-ore deposits are actually forming in Lough Neagh at the present time, although the proportion of iron is sufficient to impregnate or "petrify" wood placed in the lake.[33] If the organic matter of bogs where iron is now depositing is as high as that of tropical rivers (as seems to be suggested above), this organic content cannot be adduced as a determining factor in precipitation. More evidence is needed on this point.

Partial Explanation.—Most tropical soils, by reason of long, hot, dry seasons, are deficient in humus, and contain iron in a high state of oxidation, and therefore relatively insoluble. But regions of nearly

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[29] *Op. cit.*, pp. 63 to 97.

[30] Aschan, Ossian: Die Bedeutung der wasserlöslichen Humusstoffe für die Bildung der See-und Sumpferze, *Zeitschrift für praktische Geologie*, vol. xv, No. 2, p. 56 (Feb., 1907).

[31] Clarke, F. W.: *Op. cit.*, p. 97.

[32] *Op. cit.*, p. 85.

[33] Bischof, Gustav: *Elements of Chemical and Physical Geology*, vol. i, p. 95 (1854).

continuous heavy rainfall are covered with rank vegetation, and rapid decay yields plentiful humus. In such regions, therefore, iron may go into solution freely; but the abundant rains will cause excessive leaching of the humus, removing the dissolved iron. Thus tropical soils may, on the average, be low in humus, while the rivers carry, nevertheless, much organic matter, and hold iron in solution. The high state of oxidation of the iron as a whole, together with the extensive leaching of such iron as is soluble, and its rapid removal in waters with a high solvent power, together with the absence of small swamps giving opportunity for concentration, may possibly furnish a partial explanation of the infrequent occurrence of bog ores in the tropics.

Marine Bog Ores

In reviewing the literature of bog deposits, I have found only three references to ores now forming in marine marshes. In the tidal marshes of Nova Scotia iron sulphide is being formed.[34] No data are given concerning the extent of the deposits. Geikie[35] speaks of extensive marshes in the eastern United States in which iron is being deposited. Unfortunately he does not cite a definite locality, or give references for his authority. At Rio Tinto the mine waters are carrying iron to Huelva Bay, but no mention is made of the extent of the deposits.[36]

In none of the above descriptions do we find well-authenticated examples of typical bog ores now forming in marine marshes. Several articles on littoral sediments and sea-coast swamps[37] discuss in much detail the nature of such deposits, but nowhere in these articles is mention made of iron deposits now forming. Beck[38] concludes that aside from the deposits of iron forming at present in Huelva Bay, no really recent iron ores are known on the sea bottom.

Lake Ores

Lake ores, described as occurring chiefly in Sweden, Finland, European Russia, and Canada,[39] may be classed with bog ores, since they are

[34] Dawson, Sir William: *Geology of Nova Scotia*, p. 24.

[35] Geikie, Sir A.: *Text-Book of Geology*, 4th ed., p. 612 (1903).

[36] Beck, Richard: *The Nature of Ore Deposits*, vol. i, p. 103 (1905).

[37] Delesse, A.: Lithology of the Seas of the Old World, *Quarterly Journal of the Geological Society*, vol. xxvi, pt. 2, p. 5 (1870).

Delesse, A.: *Lithologie du Fond des Mers.*

Delesse, A.: Lithology of the Seas of the New World, *Quarterly Journal of the Geological Society*, vol. xxviii, pt. 2, p. 1 (1872).

Shaler, N. S.: Sea-Coast Swamps of the Eastern United States, *Sixth Annual Report, U. S. Geological Survey*, p. 359 (1884–85).

[38] Beck, Richard: *Loc. cit.*

[39] *Op. cit.*, p. 100.

essentially the same in distribution, method of formation, and character, and frequently show gradations into bog ores.

Summary

Bog ores are widely distributed, but occur chiefly within the colder temperate regions, and for the most part within the glaciated area. No important deposits are known to be forming in marine bogs.

Causes of the Above Distribution

The probable causes of this distribution are:

1. Swamps are more numerous in the glaciated area than elsewhere.
2. Glacial swamps are of smaller size and allow greater concentration of chalybeate waters.
3. There is abundant available iron in glacial soils.
4. Possible climatic factors involve the high state of oxidation of the iron in warmer regions, together with the lack of iron solvents in large areas of tropical soils, and abundant organic matter in many of the rivers keeping in solution the iron once dissolved, and permitting it to be carried out to sea, instead of being redeposited in swamps.

The writer believes that glaciation is probably the most important factor, operating through the favorable conditions presented by the presence of numerous small swamps, and the abundance of iron available in glacial soils.

So far as climatic factors are concerned, not enough evidence is available to warrant definite conclusions; but certain possibilities have been pointed out, and further lines of research have been suggested.

Discussion

A. C. Lawson, Berkeley, Cal.—It appears that our bog-iron ores are due in a large measure to the activity of certain iron bacteria, and a paper dealing with this subject that makes no reference to that particular agency strikes me as rather curious.

C. L. Dake, Rolla, Mo.—Mr. Lawson's criticism that this paper makes no reference to the activity of iron bacteria is scarcely true. On p. 117 I stated: "Certain bacteria precipitate iron by building the hydrated oxide into their body cells," and again, "There are many other means by which, under special conditions, iron may be precipitated in nature. Oxidation, evaporation of CO_2, and iron bacteria have been held to be the most important." Thus the activity of bacteria receives as extended mention as the action of any other agency in precipitation of iron. The paper makes no pretense of covering exhaustively the origin

of bog ores, but, as its original title suggested, refers to Some Principles Controlling the Formation and Distribution of Bog Iron Ores. Mr. Lawson seems to have entirely missed the main thesis of the paper, which attempts to establish some relation between glaciation and abundance of bog ores now forming.

Unfortunately, I have no immediate access to the literature of the subject, but my best recollection is that the data regarding iron bacteria throw no light whatever on the cause of bog ores being so much more abundant within the limits of the glaciated regions, and being reported much less frequently from the tropics.

This lack of data is suggested in the following (p. 121): "It has already been shown that bacteria aid in the precipitation of iron. Whether tropical conditions increase or decrease the activity of such bacteria is unknown."

Since I had no data showing that iron bacteria were in any way responsible for the peculiar distribution of bog ores, further reference to them in the paper cited was deemed to be extraneous to the subject.

G. H. Cox, Rolla, Mo. (communication to the Secretary*).—What appeals to me as being the most interesting and important part of the paper is the conclusion that glaciation has in general been the chief factor in the formation of bog-ore deposits. So far as I am aware, this, if true, constitutes an addition to our knowledge of the subject.

The paper with its references will be a valuable starting point for one interested in the subject, and it is to be hoped that it will increase the interest in meteoric-iron deposits so that satisfactory explanations for the solution and transportation of such quantities of the apparently almost insoluble iron oxides and the precipitation of this material as compounds varying in composition from amorphous iron hydroxide to crystalline blue hematite will be forthcoming.

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* Received Oct. 9, 1915.

Manufacture and Tests of Silica Brick for the Byproduct Coke Oven

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(San Francisco Meeting, September, 1915)

Introduction

It is rather gratifying that in the field of the manufacture of refractories, in which, as a whole, foreign practice has surpassed us, the United States stands pre-eminent in the making of silica brick. In some instances European manufacturers have refused point-blank to credit the possibility of making to such specifications as are common with us, until the actual completed shapes were shipped for inspection. But both the manufacturer and consumer in foreign lands are now frankly learning from us, and benefiting by our advanced practice in this field, as is evidenced by the orders placed here and the efforts put forth abroad to come up to the standard of American-made material.

In no metallurgical operation has the employment of silica brick spread with such leaps and bounds as in the building of plants for the carbonization of coal, particularly in the byproduct coke oven. Today, in practically all types of oven, the structure above the floor level (except the facings) is of silica, while in the recuperative and in some regenerative types, its use is extended to that portion of the oven below the floor as well.

By reason of the fact that there are a number of classes of refractories of higher silica content than the usual so-called fire-clay brick, there is sometimes a misunderstanding as to the class of material referred to as "silica" brick. We here refer only to a brick having a silica content of 94 per cent. or more and made usually from quartzite rock with a small percentage of lime as a binder. The term "silica" brick is sometimes rather loosely applied to brick from highly siliceous clays, or to a quartzite refractory of mingled fire-clay and quartz rock; but such products are not considered in this discussion.

Materials

The rock from which practically all high-grade silica brick are made is a true quartzite or metamorphosed sandstone, variously known as

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ganister, quartz, quartzite, silica rock, etc.—the largest fields, in the order of their relative importance being those of Pennsylvania, Wisconsin, Alabama and Colorado.

Pennsylvania Quartzite

Pennsylvania manufactures probably from 75 to 85 per cent. of all the silica brick made in the United States; nearly all are made from the quartzite found in the Medina and Oneida formation. While this formation extends across the State of Pennsylvania reaching into New York State on the north, and Maryland, Virginia and still farther toward the south, the quarries are practically all in Huntingdon and Blair Counties, Pennsylvania. It is in this restricted area that the quartzite is at its best. Toward the north and toward the south it becomes softer, is no longer a true quartzite, or is too much impregnated with iron. The rock is also comparatively easily accessible in the counties mentioned, enormous quantities lying loose upon the surface. The particular occurrence of this loose rock has here been brought about by the Juniata River and its tributaries, as they have cut across the formation, wearing away the softer formations beneath, and leaving the Medina overhanging and exposed. In the course of time it has been weathered and broken up by the elements into the bodies of loose rock or "floes," as they are called, which may cover an area of a thousand acres or more. The depth usually varies from 3 or 4 to as much as 20 ft. Smaller floes formed in the same way may be found along the flanks of the mountains; but they nowhere compare in size with those formed where the measures have been cut transversely.

This rock is sometimes quarried out of the solid measures, but by far the largest amount used is that taken from the bodies of loose rock, as the latter is not only cheaper to work but is of more uniform quality. The white Medina formation is not composed entirely of quartzite, but is made up principally of sandstone with intermingled strata of quartzite. Thus where the formation has been broken up, the softer sandstones unsuitable for the manufacture of silica brick have tended to weather away and disintegrate, thus leaving the larger portion of the floes consisting of quartzite. When sandstone is mixed with the quartzite in the floes it is readily detected by its appearance, as it tends to become much more rounded and worn in weathering than does the quartzite, which maintains its sharp angular edges.

The individual pieces in these floes range from those of fist-size to boulders weighing many tons. Probably 50 per cent. of the rock occurs in pieces weighing more than 500 lb. The largest floes are found at the gorges of the Juniata River near the towns of Mount Union, Mapleton, Point View, Lewistown, etc.

It is interesting to note that the Oriskany sandstone, extensively used as glass sand, is found in close proximity to the Medina and that although exceedingly pure in its composition, its softness renders it unsuitable for high-grade silica brick. It consists of a mass of minute quartz grains loosely held together, and often so friable that it can be easily worn away by the hands. On the other hand, the Medina quartzite consists of quartz crystals cemented in a mixture of the same composition and of equal hardness with the crystals. The difference in the two types of rock when ground to sufficient fineness for the manufacture of brick is obvious. The harder stone gives splintery, angular fragments; the other, rounded grains. Experience shows that a hard rock like the Medina is required for a physically strong silica brick.

The white Medina which occurs in Southern Pennsylvania near the town of Hyndman is there prominent and of a pure variety, but by no means as hard as the quartzite of Huntingdon County.

A representative analysis of the Pennsylvania quartzite is as follows:

	Per Cent.
Silica (SiO_2)	97.80
Alumina (Al_2O_3)	0.90
Ferric oxide (Fe_2O_3)	0.85
Lime (CaO)	0.10
Magnesia (MgO)	0.15
Alkalies	0.40
	100.20

Wisconsin Quartzite

This rock, much used in the manufacture of silica brick in the vicinity of Chicago, is known as the Baraboo quartzite from the town of that name where these measures appear, and near which the quarries are located. The rock occurs in high cliffs throughout this region and is especially prominent around Devil's Lake. Here glacial action has excavated a broad deep valley in the quartzite formation bounded on each side by cliffs several hundred feet in height. Their bases are covered with the loose rock split from the parent ledges. While the fragments are much larger as a whole than the weathered Medina of Pennsylvania, the quality by no means compares with the latter. Practically all the stone used in brick making is here quarried from the solid measures.

The marked peculiarity of this rock, as compared with the Pennsylvania quartzite, is its reddish color, which ranges from a light pink through a dark purplish red almost to black. It is often as red as the red Medina of Pennsylvania, a sandstone wholly unsuited for brick making. While the coloring of the Baraboo quartzite is due to iron, it is yet an exceedingly pure rock. The quartzite often becomes lighter in

color as the quarry works in from the face of the cliff, but it is the exception to find white rock.

This Baraboo quartzite, estimated as some 4,000 to 5,000 ft. in thickness, is a metamorphosed sandstone, the sedimentary origin of which is indicated by its parallel bedding planes, by ripple marks, etc. Layers of quartzite are often separated by beds of quartz schist varying from a few inches to several feet in thickness. This schistose structure is due to the admixture during deposition of clayey material, which in the subsequent folding and bending of the rock has developed a slablike or schistose structure.

The United States Geological Survey explains the formation of the quartzite as due to the enlargement of the original quartz grains and to the deposition of interstitial quartz. It is the latter that carries the iron oxides, which give the characteristic color to the rock. Regardless of its color, the formation throughout shows the physical characteristics of hardness and strength, as well as the splintered and angular fracture in crushing, identical with the Medina quartzite of Pennsylvania. A typical analysis follows:

	Per Cent.
Silica (SiO_2)	97.15
Alumina (Al_2O_3)	1.00
Ferric oxide (Fe_2O_3)	1.05
Lime (CaO)	0.10
Magnesia (MgO)	0.25
Alkalies	0.10
	99.65

Alabama Quartzite

The material used in silica brick manufactured in Birmingham, Ala., is found in the eastern part of that State. It is bluish white, and as hard and crystalline as the Wisconsin quartzite; average analyses show it to be equally pure. It occurs in lenses of varying size, in a sandstone formation. It is probably a quartzite formed by a leaching out of impurities from the parent rock followed by a crystal growth and additional deposit of silica from circulating waters. A typical analysis follows:

	Per Cent.
Silica (SiO_2)	97.70
Alumina (Al_2O_3)	0.96
Ferric oxide (Fe_2O_3)	0.80
Lime (CaO)	0.05
Magnesia (MgO)	0.30
Alkalies	0.31
	100.12

In Colorado and Montana both quartzite and sandstone are used to some extent in the manufacture of silica brick.

Method of Manufacture

Quarrying

The methods of quarrying as necessitated by the mode of occurrence of the quartzite are so obvious as to need no description. A point that will bear emphasis however, is the necessity of adequate supervision in the selection of the rock, whether it be from floes or from the solid measures. Whatever the location or occurrence of the rock, such care is essential; the easiest way to keep bad rock out of the brick is never to let it reach the plant.

Grinding

After the rock broken to one man size is received at the rear of the brick plant, it is put through some one of the familiar types of crushers, which reduces it to practically the size of a 1- to 2-in. ring. It is then elevated to storage bins from which it may be fed direct in definitely weighed quantities to the wet pans for grinding.

Into the wet pans, as the grinding proceeds, 2 per cent. of lime, by weight, is introduced as milk of lime. When the brick is burned, the lime in combination with the other elements in the mass serves to give the necessary bond or physical strength. The quality of the lime has no small effect upon the resulting characteristics of the brick. The effect of an increase in lime from 2 to 3 per cent. is not marked; but such as it is, renders the brick less able to withstand abrasion, as is indicated by rattler tests. The decrease in strength on further additions of lime up to 7 per cent. is progressive, and becomes more and more evident with each addition. There is, on the other hand, a decrease in strength at 1.5 per cent. of lime, and a marked weakness and friability as the amount of lime is still further diminished. As regards ultimate refractoriness, no marked effect is noticeable upon increasing the lime addition up to 3 per cent. From that point on, the effect is clear in a lowering of the fusion point.

The fineness to which the charge in the wet pan is to be ground depends upon the nature of the brick to be made from it and is, of course, controlled by the time during which the charge is allowed to remain in the pan. In general, one charge, containing material for 200 brick, will be ground at the rate of approximately four pans per hour for standard 9-in. b ick, while to obtain the fineness requisite for the intricate and difficult shape-work necessary in byproduct coke-oven construction, but two pans per hour may be possible. The rock is exceedingly hard, and a comparatively slight increase in the fineness of grind is accompanied by a seemingly disproportionate lowering of grinding capacity, and increase in wear on the pans.

The actual differences in grind, however, are much smaller than is often imagined. Figs. 1 and 2 are photomicrographs with a magnification of 2 diameters, the ground material showing on a background of cross-section paper with 0.1-in. spacing. The specimens are from pieces of green or unburned silica brick of equal weight, crushed down after drying. Fig. 2 indicates typical fine-ground shape-work, while Fig. 1 is that of average standard 9-in. brick.

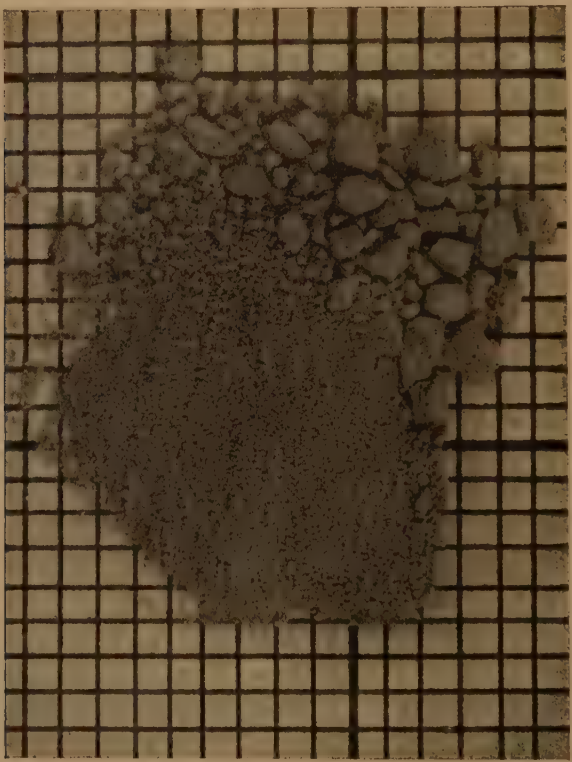

Fig. 1.—Material from Unburned Average Standard 9-in. Brick. Magnification 2 Diameters.

It will be plainly apparent from the photomicrographs that the difference lies not so much in the actual size of either the coarse or the fine grains as in the relative proportion of fines. In the finer-ground brick the larger proportion of these is clearly evident. There can be no question that the general effect of increasing the fineness of grind is to improve the abrasive resistance of the brick, if it is properly burned. In general the requirements as to grind contained in typical coke-oven specifications have worked no hardship upon the manufacturer; for the present-day standard of workmanship on byproduct oven shapes is such

as to "carry its own grind with it," if the expression may be used. To secure the accurate filling out of the shapes, and the clean-cut accuracy everywhere essential, the grind is actually much finer than is requisite to satisfy the usual specifications as to that particular feature.

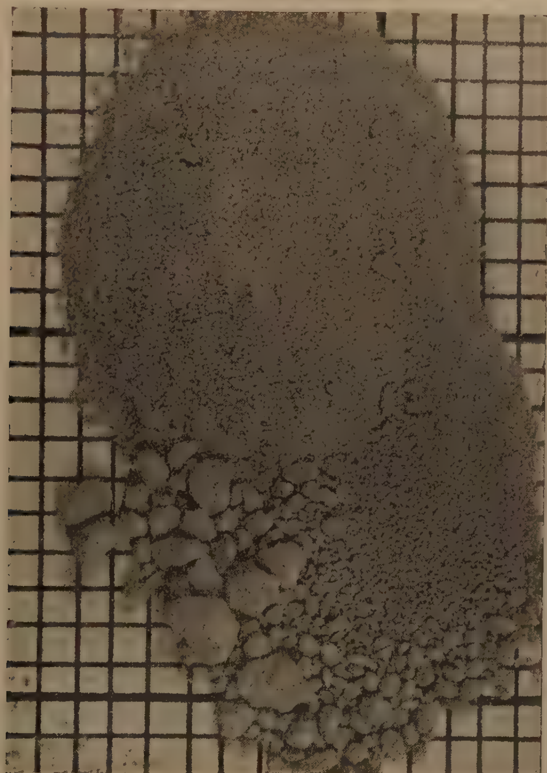

FIG. 2.—SAMPLE SHOWING MATERIAL USED IN TYPICAL FINE GROUND SHAPE-WORK MAGNIFICATION 2 DIAMETERS.

Molding

The ground silica material as ready to mold is of about the consistency of damp sand, but probably because of the extreme angularity of its grains has less flow as it is manipulated. As a consequence, the molds must be filled by pounding with heavy plank beaters covered with metal, and all corners and shapes of irregular outline must be rammed in by hand, much as sand molds are made, but with much heavier blows. Unequal ramming, which leaves too great a variation in the density of the green brick, will give unequal expansion in the kiln. It is by no means uncommon to lose hundreds of a new and difficult shape through such fire-cracking. To get proper molding and ramming requires the strictest supervision.

The wear on the molds due both to the attrition of the grains and the pounding is exceptionally severe. In view of the extreme accuracy required, steel molds are used exclusively for all high-grade coke-oven work. These molds are often decidedly complicated, and always expensive, as "loose sides" must be used throughout. Ground silica has, as molded, almost no physical strength, and can be taken from the mold only by allowing the sides to come with it, after which the sides are stripped.

After molding, the shapes are placed on pallets, on which the brick are dried either on the steam-heated hot floor or on rack cars in some form of tunnel dryer (Fig. 3). The brick are then ready to be set in the kilns. Although setting is a most important feature in the manufacture, no attempt will be made to discuss it here.

FIG. 3.—TYPICAL SECTION OF HOT FLOOR, SHOWING COKE OVEN "GUN" BRICK-SHAPES IN PROCESS OF DRYING.

Burning

The vast majority of silica brick are burned with soft coal in the familiar type of round, down-draft kiln, varying in diameter from 30 ft. upward. Although practice may vary somewhat, and much depends on the size of the kiln, the following schedule of burning may be considered as fairly normal. Assume that the setting is complete and the doors are sealed. Fire is immediately started, and continued slowly and with slight acceleration for from 9 to 12 days, during the last 2 or 3 of which the final temperature should be practically attained. The kiln is then held at the required temperature, say at 26 cone (1,650° C., or

3,002° F.) directly in front of the firebox, for another day, after which the fireboxes may be sealed and allowed to remain thus for approximately 24 hours more. Cooling will require about as long as the burning proper; but this varies somewhat with atmospheric conditions.

It is obviously difficult to take temperatures in the interior portions of the kiln except by cones; but there are often most perplexing variations in this method. There are apparently certain conditions in the atmosphere of a burning silica kiln which alter the readings of standard cones to no small degree. Cones which give concordant results, week in and week out, in clay kilns are often decidedly erratic in registering the temperatures of silica kilns. It is by no means uncommon to witness cones of higher number go down, while next to them those lower in the series are either but slightly bent or apparently unaffected.

Effect of Burning on Composition and Expansion

In any discussion of the desirable qualities of silica brick there is frequently expressed the necessity that "all the expansion shall be burned out of them." This brings out the following questions: What is the nature of this expansion; what produces it; what is its extent, and how fully can it be controlled?

That the terms used in the discussion may be clear, let it be premised that all silica brick are of course made in molds smaller than the proposed size of the finished or burned brick, and that under the influence of heat a permanent expansion occurs, swelling the brick from the mold or "green" brick size to the required or "burned" size. It is desired that this permanent expansion occurring during burning be completed so as to preclude further permanent expansion in subsequent use.

Now, in every burned silica brick expansion will occur on heating—a true thermal expansion, identical in its general character with that incident to the heating of practically all bodies. Such thermal or temporary expansion will disappear upon cooling, and being the expression of the coefficient of expansion of the silica it is unavoidable. This thermal or temporary expansion, occurring upon heating a burned silica brick, is not to be confused with the permanent expansion produced in the body of the brick during its manufacture. It is argued that if a silica brick is not properly burned, then, upon reheating in actual coke-oven practice, there will be not only a normal thermal expansion but an additional expansion of indeterminate amount which will be permanent. It is essential in the construction of the coke ovens that the expansion, whatever its nature, be as small as possible, and that, whatever it is to be, it be known and adequately cared for.

It will be recalled that the silica brick under consideration is formed by adding 2 per cent. of lime to quartzite rock of the following composition, say:

	Per Cent.
Silica (SiO_2)	97.80
Alumina (Al_2O_3)	0.90
Ferric oxide (Fe_2O_3)	0.85
Lime (CaO)	0.10
Magnesia (MgO)	0.15
Alkalies	0.40
	100.20

The representative silica brick produced may be assumed to have the following composition:

	Per Cent.
Silica (SiO_2)	96.25
Alumina (Al_2O_3)	0.88
Ferric oxide (Fe_2O_3)	0.79
Lime (CaO)	1.80
Magnesia (MgO)	0.14
Alkalies	0.39
	100.25

What are the changes which occur as the siliceous mass is burned?

Certain of these are obvious, such for instance as the formation of a bond of one or more of the lime-silica compounds such as the meta-silicate CaO, SiO_2 or the ortho-silicate $2CaO, SiO_2$. With the exception of the lime present, the ferric oxide of iron and the alumina, though small in quantity, are the preponderating impurities and there is undoubtedly formed a certain amount of silicate of iron as well as compounds of the lime-alumina series. Whereas the lowest melting point incident to any eutectic of the lime-silica system is 1,436° C. (2,617° F.) and of the alumina-silica series is 1,610° C. (2,930° F.), while that of the lime-alumina system is 1,400° C. (2,552° F.), some of the eutectics of the ternary system CaO, Al_2O_3, SiO_2 melt at approximately 1,170° C. (2,138° F.) and are thus among the first of the flux compounds resulting.

No less important, though less recognized, changes are occurring in the brick as it is heated. The evidence of the change, regardless of its reason, is easily indicated by the most casual measurement when brick are drawn from the kiln from time to time, as higher temperatures are reached. The final expansion will be very close to ⅜ in. per foot.

The theory of this expansion is that a change occurs in the crystalline form of the silica, an inversion from the quartz crystal to the crystalline form of crystobalite. The intermediate crystalline form of tridymite does not apparently occur here. Since these various crystal forms have varying specific gravities the volumes accordingly vary.

The valuable work done by Day and Shepherd[1], and particularly by Fenner,[2] gives the results in a large field of laboratory experiment,

[1] The Lime-Silica Series of Minerals, *American Journal of Science*, 4th series, vol. xx, No. 130, pp. 265 to 302 (1906).

[2] The Stability Relations of the Silica Minerals. *American Journal of Science*, 4th series, vol. xxxvi, No. 314 (1913).

which make possible a fair understanding of what occurs in the actual burning of the silica brick. Fenner's work was to establish among other things the relations between the silica minerals quartz, tridymite, and cristobalite, any of which may be converted into any other of these by appropriate heat treatment. For the purposes of this discussion it is unnecessary to touch upon the alpha and beta phases of these minerals. In substance, the results under the conditions imposed by the experiments were that powdered quartz, if heated, remains quartz until a temperature of 870° C. (1,598° F.) is reached, at which inversion point there begins a slow change to tridymite. If heat is still applied, there is no further change from tridymite until 1,470° C. (2,678° F.) is reached, when inversion to the form of cristobalite has its beginning. There is no further change until the cristobalite melts into a glass. Thus the stable phase of quartz (quartzite rock as applied to silica-brick manufacture) extends to 870° C. (1,598° F.). Between 870° C. (1,598° F.) and 1,470° C. (2,678° F.) the stable phase is tridymite, while above 1,470° C. (2,678° F.) cristobalite is stable.

As a whole it must be carefully noted that the changes from one mineral to another were exceedingly slow, particularly near the inversion points. So slow in fact were the alterations, that the melts were made with a large excess of sodium tungstate which acted as a flux or catalyzer. The powdered quartz or whatever might be the silica mineral under study was stirred into the molten bath of flux and the platinum container was then heated in a suitable furnace. This flux in no way entered into chemical combination with the silica but only aided and quickened its action, apparently furnishing a fluid medium through which the molecular rearrangement was facilitated. In spite of this, the slowness of the inversion, often requiring many hours, will bear emphasis in drawing conclusions regarding any parallelism between these experiments and the burning of brick.

A particularly noteworthy fact is that in the absence of a flux or catalyzer, the inversion from quartz to tridymite does not occur, but the inversion is direct to cristobalite, and there is a shift in the inversion point to as low as 1,250° C. (2,282° F.) although the change at this temperature is slow.

This entire absence of tridymite in quartz heated well above 870° C. (1,598° F.) the quartz-tridymite inversion point, although difficult of explanation, is apparently proven. As indicative of the time element requisite for such an inversion, in one experiment fine-ground quartz heated 108 hr. at 1,250° C. (2,282° F.) without flux gave but a small percentage of inversion while with 90 hr. at 1,360°C. (2,570° F.) approximately two-thirds became cristobalite.

In considering these changes in the crystalline form of silica, some of the characteristic properties of the three will not only make clearer the method of distinguishing them, but also throw some light on the reasons for the effects these inversions exhibit.

Quartz has a specific gravity of 2.65, tridymite of 2.27, and cristobalite of 2.33. Each has its characteristic crystal form, the quartz being usually in the familiar hexagonal pyramids, prisms or combinations of these. Tridymite often occurs in thin tablets or plates hexagonal in shape. Cristobalite is exceedingly irregular in the development of its crystal form but often develops octahedral caps along the closely interlacing axes of elongation.

It is the difference between the indices of refraction of these minerals, however, that furnishes the greatest aid in distinguishing the one from the other.

Tests.—As a starting point from which to work in determining the effect of burning, a set of sample silica brick were taken from one kiln, all from the same zone and from a kiln of absolutely normal burn. In this instance they were first burned at a point about 8 ft. from the wall and 6 ft. from the bottom. This sample was divided into three parts, one to be kept unaltered, one to be reburned under the same conditions, and a third to have two reburns. In addition to this, pieces of quartz rock received one and two burns respectively, with the view of obtaining the same data.

The primary object was to determine, if possible, how far the inversion from quartz to cristobalite had progressed under the successive conditions. Each burn was made as nearly uniform as practicable, and in each succeeding burn the effort was made to maintain identical conditions. At that part of the kiln where these brick were burned the temperature reached 1,540° C. (2,804° F.), and they were subjected to that heat for approximately 40 hr.

* In general, the chief optical constant serving to distinguish cristobalite on the one hand from quartzite and calcium silicate on the other was the index of refraction. The slow ray of quartz has a refractive index of 1.544 while the index of the fast ray of cristobalite is about 1.485. The immersion method was used to determine the indices of the grains, the powdered brick being immersed in a liquid of index 1.51. By this method it was possible to distinguish between the higher-refracting quartz and calcium silicate and the lower-refracting cristobalite.

With the aid of a *camera lucida*, the fields were sketched on very thin tracing paper. The grains, as represented on the paper, were then cut out and weighed for each constituent. Aside from the fact that the composition of different fields may vary, other errors are entailed in the sketching, cutting out and weighing so that a variation of 5 per cent. is justifiable. In these determinations the fields of the same specimen

* The heartiest appreciation is here expressed to A. A. Klein, Petrographer in the U. S. Bureau of Standards at Pittsburgh, for his courtesy in making these determinations that they might be concluded in time for presentation here. The quality of Mr. Klein's work needs no comment.

checked to within 5 per cent. maximum and 1 per cent. minimum error.

Results.—The tabulated data are as follows:

Specimen	Field	Quartz and Calcium Silicate, Per Cent.	Average, Per Cent.	Cristobalite, Per Cent.	Average, Per Cent.
Lot 1	1	24.87	....	75.13	....
(1st burn)	2	20.42	22.65	79.58	77.35
Lot 2	1	14.31	....	85.69	....
(2nd burn)	2	19.96	17.13	80.04	82.87
Lot 3	1	17.44	....	82.56	....
(3rd burn)	2	14.60	16.02	85.40	83.98
Quartz rock	1	50.08	....	49.92	....
(1st burn)	2	51.05	50.57	48.95	48.95
Quartz rock	1	32.70	....	67.30	....
(2nd burn)	2	30.06	31.38	69.94	68.62

A number of interesting deductions may be made even from these preliminary tests. The maximum change takes place at the first burning, followed by a further appreciable inversion at the second, and practically no change at the third.

The grain structure of the reburned silica brick may throw some light on this subject. Here the grains may be seen to be essentially cristobalite, with the quartz occurring as an inclusion. The last stages of the inversion are apparently exceedingly slow within the protecting envelope.

The smaller percentage of inversion in the case of the quartz rock as compared with the brick is very noticeable, and is undoubtedly due for the most part to the grinding of the quartz in the manufacture of the brick. This is wholly in line with Fenner's laboratory experiments, in which he found the influence of comparative grinds to be slight if the silica mineral were heated with a flux, but to be marked if the material were heated without a flux. Fine grinding greatly hastened the inversion. The greater freedom of movement in expansion as compared with the exceedingly dense quartz body is probably the controlling factor, rather than any difference in the penetration of the heat.

The part played in the inversion (if any part is played) by the lime and its compounds present, is of much interest. It appears logical that the greater fluidity conferred by the fluxes present should have an influence; but it must be borne in mind that, as compared with previous experimental work on silica minerals, the amount of flux present in the brick is exceedingly small, and that definite chemical compounds are formed in combination with the brick body. In Fenner's work the sodic tungstate formed no such chemical compounds, and remained unchanged at the conclusion.

The fact moreover that no tridymite was determinable in either the burned brick or the calcined quartz is explained by the presence of the

negligible amount of flux. This would favor the direct inversion of quartz to cristobalite.

Thus far, we have assumed the changes in crystal form to be desirable, because of their effect on the expansion of the material.

Effect of Burning on Physical Strength

It is also of interest to note the effect of the reburning on the physical strength of the brick. To determine this, the following tests were run:

Cross-Breaking or Modulus of Rupture Test

Number of Burns	Mark	Breadth of Brick, Inches	Depth of Brick, Inches	Weight Causing Fracture of Brick, Pounds	Length between Supports, Inches	Modulus of Rupture,	Average, Pounds
1	9 A	2.46	4.40	3,550	6	671	....
1	9 A	2.42	4.40	3,000	6	576	624
2	9 B	2.50	4.50	4,140	6	730	....
2	9 B	2.50	4.50	5,000	6	889	809
3	9 C	2.06	4.50	4,500	6	971	....
3	9 C	1.96	4.50	4,550	6	1,031	1,001

Crushing-Test: Whole Brick Crushed Flat

Number of Burns	Mark	Dimensions, Inches	Area, Square Inches	Load, Pounds	Crushing Strength per Square Inch, Pounds	Average, Pounds
1	9 A 2	9.00 by 4.40 by 1.96	39.60	176,800	4,465	....
1	9 A 2	9.04 by 4.44 by 2.04	40.14	167,000	4,161	4,313
2	9 B 2	9.10 by 4.46 by 2.02	40.58	186,500	4,596	....
2	9 B 2	9.10 by 4.40 by 2.06	40.04	168,000	4,196	4,396
3	9 C 2	9.06 by 4.42 by 2.00	40.04	195,000	4,870	....
3	9 C 2	9.12 by 4.46 by 2.06	40.68	168,000	4,130	4,500

The increased strength of the brick is evident.

Conclusions

In considering whether or not all the expansion possible is burned out of the brick, it is necessary to determine first the limits and then how closely they are approached.

Day, Sosman and Hostetter say[3] that the true specific volume of cristobalite as compared with quartz at 20° C. is 13.4 per cent. greater.

[3] The Determination of Mineral and Rock Densities at High Temperatures. *American Journal of Science*, 4th series, vol. xxxvii, No. 217, pp. 1 to 39 (1914).

Assuming a green brick to be wholly pure quartz (100 per cent. SiO_2), and the brick burned from it wholly cristobalite, the volume increase would then be 13.4 per cent. if the relative porosity of the body be disregarded. This theoretically possible expansion is diminished by the effect of the impurities present in the actual brick, and by the resulting compounds formed. For instance, 4 per cent. of impurities would produce a much larger proportion of compounds to which this expansion phenomenon does not apply.

An expansion of, say, 3/8 to 7/16 in. per foot, to which byproduct coke-oven silica is burned, thus provides for somewhat more than 13 per cent. increase in volume.

It must be borne in mind, however, that by quick burning the brick may be easily expanded by double or treble the normal amount; but sound brick do not result. This excessive expansion is due to minute fissures opened up in the body of the brick by a too sudden rise in temperature during burning. In the early stages of firing the heat must be raised very slowly. In general the conclusion is justified that the present good commercial practice carries the results of the quartz-cristobalite inversion to the greatest degree economically possible and that the measure of the expansion of the brick affords a closely approximate measure of the inversion—providing only that the brick are sound.

The short series of tests outlined above are given with full realization that, while indicative, they are by no means conclusive. They were but the preliminary runs, necessarily made somewhat in haste, but giving data which permit intelligent plans to be made for the more comprehensive tests now under way. There is much to be learned as to the effect of both time and temperature as well as the influence of pressure, which may produce a shifting of some of the inversion points. Judging from the work thus far done, it is impossible to expect the full inversion of quartz to cristobalite under any practical operating conditions of manufacture.

Discussion

J. W. Richards, South Bethlehem, Pa.—When speaking about coke-oven brick, the first thing that comes to my mind is how important it would be if it were possible to get a brick of better heat conductivity. The heat for coking coal has to go through the thickness of the brick. The silica brick used has low heat conductivity as compared with some other refractories; magnesite brick probably has four times its heat conductivity. The coke oven needs bricks of high heat conductivity. Any process which would increase the heat conductivity of silica brick would undoubtedly shorten the coking process.

William A. Williams, San Francisco, Cal. On p. 138, the tests of bricks that have been burned once, twice and three times, and the width

of the brick in inches, is given. From the first burn to the second there seems to be an expansion, and from the second to the third a remarkable contraction.

KENNETH SEAVER.—Part of those tests were made on what we term 9-in. straights and part were split bricks, and not on the same brick. It is confusing the way it is expressed in the paper.

J. W. RICHARDS.—Speaking now on the general topic of suitability of firebrick and silica brick for coke-oven construction, I have here a sample of fire-clay brick from a coke oven which has melted down because it was subjected to a higher temperature than it could stand. It is interesting to note that the upper part of the brick, close to where it was melted, shows that the mortar used was not melted. In other words, the brick was more fusible than the mortar. I have also some samples of silica brick which came from the same ovens and were subjected to an exceptionally high

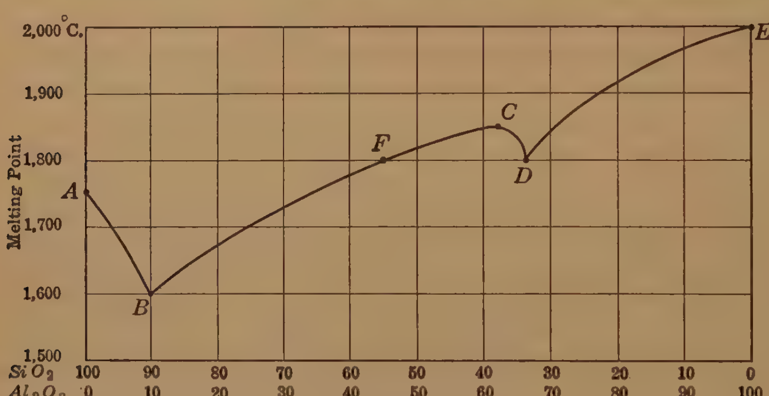

FIG. 1.

temperature. There is no sign of these bricks melting, but the mortar between the bricks, fused, attacked the bricks, and flowed out, the bricks themselves being unfused. This behavior is quite comprehensible if we refer to the melting-point diagram of SiO_2–Al_2O_3 mixtures (Fig. 1), where *A* represents the fusion point of pure SiO_2 (1,750° C.); *B* the eutectic containing 10 per cent. Al_2O_3 (1,600°); *F* the composition of kaolin, 46 per cent. Al_2O_3 (1,800°); *C* the compound $Al_2O_3SiO_2$ (1,850°); *D* the second eutectic, at 67 per cent. Al_2O_3 (1,800°); and *E* pure Al_2O_3 (2,000°).

A study of this diagram will show that fire clays containing 10 to 33 per cent. of Al_2O_3 (not considering the influence of impurities) have melting points below that of pure silica. If such clays are made into firebrick they may melt at temperatures of 1,600° to 1,750° C. (lower if they contain impurities), but will not be fluxed by clays with a higher content of Al_2O_3. It thus appears wise to use as mortar, with firebricks, a clay with a higher Al_2O_3 content than the brick; mortar clay with a

lower Al_2O_3 content than the firebrick would flux the brick, but with a higher Al_2O_3 content than the brick should be without fluxing action. The latter appears to be the condition in the sample shown, where the firebrick has melted but the mortar is unchanged.

On the other hand, a silica mortar, with less than 10 per cent. of alumina, would be the worst possible mortar to use with a fire-clay brick.

Turning to the question of mortar for silica brick, it is evident from the diagram that any fire clay from *B* upward toward *F* is an improper material to use between silica bricks, as mortar, because it must tend to form the eutectic *B*, by fluxing the bricks; also, the further toward *F* it is in composition, the greater will be its power to flux silica and therefore the more disastrous the results. In other words, the better fire clay it is, the worse mortar it is, for silica bricks. On the other hand, the nearer to *B* the mortar is, the less will be the amount of fluxing action it can have on silica bricks. If mortar of composition *B* is used between silica bricks, it can only attack silica by forming a compound between *B* and *A*, and thus by automatically *raising* its melting point. Clay of the composition *B*, or some material between *B* and *A*, is the best for using between silica bricks, as its fluxing power on silica diminishes in going from *B* to *A*, but increases if its alumina rises above the amount in *B*. The specimen exhibited, which showed the mortar run entirely away, in places, from between the silica bricks, at the same time strongly fluxing the surface of the bricks, but at a temperature evidently well below the melting point of the bricks themselves, is a good example of the action of a high-alumina fire-clay mortar in fluxing silica bricks. With silica bricks, the higher the alumina in the fire clay used as mortar (above 10 per cent.), and therefore the higher its melting point in and of itself, the worse fluxing action will it have on the bricks.

A corollary of these observations is that fire clay for a mortar should be tested, before using, in combination with the bricks with which it is to be used. It is of no use to say that the mortar is first class fire clay. with a melting point of 1,750° or 1,800° C. This property may be an indication of its total unsuitability for use with silica bricks. For silica bricks, the best mortars must be the fire clays lowest in alumina (not considering impurities), and a silica wash with just enough clay with it to make it stick, should be the best mortar.

F. G. Breyer, Palmerton, Pa.—I note that the fire-clay brick fused before the silica brick. This does not seem quite comprehensible to me in connection with the diagram which Professor Richards has shown. I should like to ask whether 1,800° C. is good coke-oven practice; it seems to me rather high. Perhaps the failure of the brick was due to spalling. In zinc work the fire-clay bricks give the best results and the greater percentage of the failures are not due to fusion of the brick but to spalling.

J. W. RICHARDS.—Fire-clay brick is not necessarily of the composition of kaolin, but may be much lower in alumina, besides containing impurities, such as lime, iron oxide or alkalies.

F. G. BREYER.—The fire-clay brick you show might have been a cheap brick. If it had been a good fire-clay mixture it might have had a higher fusion point. I think the failure must have been due to the breaking off of the brick because a fire-clay brick with fire clay between has a higher fusion point than silica brick, even though you use no alumina mortar but use a straight silica mortar between.

J. W. RICHARDS.—The other horn of the dilemna is that you do not have a good fire-clay brick.

F. G. BREYER.—That is possible, but if you start out on the basis of having a good fire-clay brick I cannot see that the temperature is the real point at issue.

ALBERT SAUVEUR, Cambridge, Mass.—The diagram refers to ideal fire clay, not commercial fire clay.

F. G. BREYER.—We have fire clay melting between 1,700 and 1,800° C. Our requirement is 1,630° but we frequently get over 1,700°.

C. G. ATWATER, New York, N. Y.—I wish to say that in my opinion it was not spalling that caused the failure of the brick in the case under consideration. It was unquestionably fusion. Coke-oven practice does not require temperatures in the oven chamber as high as those referred to, since the burning of gas is done not in the oven but in the flue. The oven is not so hot but the flue bricks must stand as high a temperature as can be maintained. The inability of any fire-clay brick obtainable at a price lower than that of silica brick forced the construction of coke ovens to pass from the quartzite or clay composition brick to the straight silica brick. It was not desired, but it was the only thing to do because the quartzite and clay brick proved insufficient. It was a question of fusing and not of spalling, however.

F. G. BREYER.—The question of fusing depends also on the possibilities of infiltration of gases. Perhaps alkali was brought in in this way and lowered the fusion point of the clay brick more than that of the silica brick. But from the diagram shown by Professor Richards it would appear that no advantage is obtained by changing from alumina brick to silica brick.

C. G. ATWATER.—Is alumina brick of the necessary purity commercially possible?

F. G. BREYER.—Certainly. Such brick would not fuse as easily as silica brick and would melt above 1,700° C.

C. G. Atwater.—As regards the possibility of alkaline gases aiding fusion, I might say that the fusion does not occur in the oven, but in the flue, from which are excluded the gases being liberated from the coal.

William Hutton Blauvelt.—The use of brick in a coke oven demands much more severe specifications than in most other metallurgical work. In a block of coke ovens we have a very large structure with many forces of expansion and contraction working one against the other. We have considerable height of structure and also heavy loads in some parts. In the work of the Joint Refractories Committee, of which I am a member, we found that the loads carried by coke-oven bricks are among the highest loads that exist under high-temperature conditions.

The stiffness of silica brick at high temperature and under heavy load is one of the important reasons why silica brick has been largely adopted in coke-oven work. Mr. Breyer says truly that fire-clay brick of high quality will stand very high temperatures without fusion, but at those high temperatures clay bricks become soft and are distorted under load, even though they do not melt. Silica brick does not do this.

There are two kinds of shrinkage in brick: the first is the shrinkage in burning, found only in clay brick. Silica brick expands on burning. This is permanent. Secondly, both clay and silica brick expand and contract like metals, temporarily, on heating and cooling. Clay brick will contract in use under high temperatures, thus tending to make the structure leaky. Silica brick expands on heating, thus tending to make a structure tighter. But with the latter brick careful provision must be made to allow for this expansion. In byproduct coke-oven practice in Europe they have endeavored to develop a brick having neither expansion nor contraction by using a clay brick high in silica, called quartzite.

I think that the general metallurgical public does not realize how severe are the conditions under which bricks are used in modern byproduct coke-oven practice and the difficult conditions met by the brickmakers in meeting some of the specifications. The manufacturers of silica brick in America should be complimented on the way they have worked out these problems.

Referring to what has been said about mortar for high-temperature brick, it has been my practice to test it as nearly as possible under working conditions. The mortar is "buttered" between two bricks in the ordinary manner of making a joint. If the mortar stands up in the test oven as it should, under these conditions of test, it can be relied upon to give satisfaction in the wall. In practice we use mortar of approximately the same composition as the brick but made a little more fusible by the addition of some salt or perhaps some plastic clay to make it work more easily under the trowel.

Rotary Kilns for Desulphurization and Agglomeration

BY SAMUEL E. DOAK, E. M., PHILADELPHIA, PA.

(San Francisco Meeting, September, 1915)

The utilization of rotary kilns, of the well-known cement type, for the preparation of iron ores for the blast furnace, has become of considerable economic importance within the past 10 years in certain localities, notably those where the native ores are lean. Such kilns may be used simply for the agglomeration of ores too fine to be charged into the blast furnace (*i.e.*, ores whose loss in the dust catchers is excessive, and whose fineness is detrimental to the working of the furnace), or for the desulphurization of lump and fine ores the sulphur content of which does not exceed 6 or 7 per cent. The desulphurizing action can be combined with that of agglomeration provided the kiln is sufficiently large. Other processes have been developed for the same purpose, but while I am not able to draw a comparison between them by reason of lack of sufficient data, it is safe to say that the kiln, if properly dimensioned for the character of ore to be treated, will compare favorably in its results, both as to economy and quality of product, with any other process. It is not, however, as well adapted for the treatment of flue dust as is a sintering operation, on account of the large percentage of fine coke contained therein. Any combustible entering the kiln at its feed end renders the process very much more expensive and unsatisfactory on account of the difficulty of controlling the temperature, also because of the rapidity with which long "rings" form, due to the increased heat at the cooler end of the kiln.

The greatest value of the kiln at present is in the treatment of pyrites cinder; that is, the residue from pyrites ore after the extraction of the greater part of the sulphur. This cinder, while high in iron content, has too much sulphur remaining in it to be allowed to form a large part of the blast-furnace burden, without further treatment. It will range in metallic iron from 42 per cent., representing a very low-grade cinder from a domestic pyrites, to 64 per cent. of the residue from the best Spanish ore, and the corresponding silica content will vary from the maximum of about 25 per cent. to 1½ per cent. The residual sulphur depends largely on the type of burner used in the original desulphurization. In general, the leanest original ores retain the greatest amounts of sulphur. In modern practice the percentage of sulphur in the cinder is rarely as great as six, and seldom lower than one and one-half.

Average analyses for iron, silica, and sulphur of the cinders that have come under my personal observation are as follows:

	Iron, Per Cent.	Silica, Per Cent.	Sulphur, Per Cent.
Virginia fines	48	18	4.2
Virginia lump	54	12	5.0
Spanish fines	58	8	2.5
Spanish lump	63	2	2.0

In addition, nearly all cinders contain zinc, copper (frequently in sufficient amounts to be extracted commercially as a by-product), calcium, magnesium, etc., and sometimes lead. The total of these accompanying elements, however, is usually well under 6 per cent.

The process is a simple roasting, with the addition in the case of fine ores of a nodulizing or agglomerating operation, performed near the discharge end of the kiln, at a higher temperature than is necessary for the desulphurization. The sulphur is combined in the ore as sulphide and sulphate of the various bases, and if the kiln is not forced over its capacity there is no difficulty in reducing it to less than 0.05 per cent. of the finished product. As far as the desulphurizing action is concerned, the principal caution to be observed is not to allow the kiln to get too hot at the back end. To get a dead roast the operation should be very gradual, and the heat should range from a barely perceptible dull red at the feed end to a bright yellow at the discharge end. When the ore is too hot in the center of the kiln, there seems to be a tendency to hinder the desulphurization, probably on account of an almost imperceptible glazing of the particles of cinder, that renders them impervious. For this reason it is important that the roasting should be complete before the nodulizing begins, since for the success of the latter operation the ore should be sufficiently pasty to ball up. This pastiness, however, should never be carried to the point where there is any flow of the cinder, and in fact it should never be visible in the kiln. A bright yellow heat is sufficient for the nodulizing, and it is well to keep the zone of agglomeration as near the discharge end of the kiln as possible, 15 ft. being sufficient length for the operation in any kiln not more than 100 ft. in length. For this reason a short flame is very desirable.

Oxygen necessary for the desulphurization is furnished either by stack draft at the feed end, by a blower at the discharge end (in connection with the fuel supply), or by both. The height of the stack is determined more by the necessity of raising the sulphur fumes above the height where they are obnoxious to people, and harmful to vegetation, than by the oxygen requirements of the process, as the slope of the kiln together with the draft induced by the coal blower is sufficient in small kilns to furnish the required amount of air without a stack. A kiln 60 by 5 ft. was run for several months without any stack, with excellent results, the only artificial draft being supplied with the powdered coal by an Aero pulverizer.

This same kiln on an 80 by 4 ft. stack gave incomplete desulphurization and an excessive amount of "ring" trouble, due to too high a temperature at the feed end.

At present the rotary kiln owes what popularity it possesses to its nodulizing feature, in spite of the fact that it is a cheap and efficient roaster, and that it is much more successful, as a roaster, working on coarse than on fine ore. As I remarked above, the sulphur should be practically eliminated from the ore before nodulizing begins. The most satisfactory way of insuring this would be to separate the two processes by using two kilns, but that would be impracticable except in a very large plant, and as a matter of fact the whole operation can be done very well in one kiln provided it is not too small. The greatest advantage of separating the processes would be that "ringing" could be almost entirely eliminated in the desulphurizing kilns, and confined to the nodulizing kilns, which could be made short and of large diameter and great capacity.

It has been the custom arbitrarily to connect a certain diameter with a certain length of kiln. A 60-ft. kiln is usually 5 ft. in diameter; 80- and 85-ft. kilns are 6 or 7 ft. in diameter, and a 100-ft. kiln is ordinarily 8 ft. in diameter, but it would frequently be more economical to proportion the kiln in a less haphazard way than by accepting the builder's standard designs. The two determining factors should be the output desired, and the quality of ore to be roasted. The output is dependent on the dimensions of a cross-section of the bed of ore that can be carried in the kiln, the slope of the kiln, and its speed of revolution. The slope of the kiln, however, can be left out of consideration. Good practice places it at ¾ in. to the foot, and increased or decreased output can be better obtained by changing either of the other variables than by changing this. Speed control, too, is a feature of the design of every kiln now, so that this factor may be adjusted to the most satisfactory point after the kiln is in operation. The question of output, then, so far as the kiln design is concerned, is determined by the width and depth of the ore bed. In general, the latter can be increased with the length of the kiln, since the length of time it takes the ore to pass through the kiln will be greater as the kiln grows longer, all other things being equal, but the increased capacity due to the length will not be very great, since, without changing the diameter, the ore bed cannot be deepened to any great extent without harming the roasting process, for which a continual and complete stirring and overturning of the ore bed is necessary. Too great a depth of bed prevents this.

An increase in the diameter of the kiln, on the other hand, increases the possibilities of the output considerably, by allowing the cross-section to be made much greater with a slight increase in depth, giving the desired effect of a greater ratio of surface of ore exposed to the air in comparison with the area of the cross-section. It would seem reasonable, then,

to base the length of the kiln entirely upon the time necessary for desulphurization and agglomeration, and to control the output by the diameter.

The question of the speed of progression of the ore in the kiln, upon which, together with the kiln length, depends the time available for treatment, cannot be accurately determined beforehand, as it is controlled by too many variables. The size of the ore particles, and their shape, the depth of bed, condition of the lining, and the size and shape of the ring all have an effect on the speed of progression of the charge. Fortunately, however, we know from actual practice that any type of cinder can be treated satisfactorily in kilns 80 to 100 ft. in length, turning at a speed of from one revolution in 60 sec. to one in 90 sec. The greater length is safer in treating all fine, high-sulphur cinders, since it reduces greatly the chances of a poor product through careless supervision.

The tonnage output of a kiln varies greatly with the kind of cinder charged. A kiln 60 by 5 ft. with 4½-in. lining will treat 1,500 tons a month of low-sulphur lump ore, while on high-sulphur fines, it will with difficulty take care of 750 tons, and frequently even at this low tonnage a large percentage of the product will not be nodulized. A kiln 85 by 7 ft. will handle 3,000 tons of high-grade cinder per month, and about 1,500 tons of the poorest grades, while a kiln 100 by 8 ft. will vary from 4,500 to 2,500 tons with the quality of cinder treated. A kiln 85 by 8 ft. should handle almost as much as the one 100 ft. long, but at a greater fuel cost. In general, the larger kiln will have the lower fuel cost and the fine domestic ores will need less coal, on account of their higher sulphur content. On a 60 by 5 ft. kiln, the fuel consumption averaged for several months about 12 per cent. by weight of the weight of roasted ore.

The mechanical difficulties that were so common when rotary kilns were first used for the treatment of iron ores have been practically eliminated. The most important things to be observed to insure continuous operation are, to make the kiln sufficiently strong (reinforcing it especially very heavily under the tires), and to keep the operating machinery well protected from the dust. The tires should always be kept tight to the shell, as if any play is allowed the bolt holes will gradually become enlarged, and the kiln will bear down hill so strongly that it will frequently break its thrust bearings. At least an inch of additional metal should be put around the shell under the tire.

The one great obstacle now in the way of continuous operation is the formation of "rings," caused by the cinder becoming hot enough to stick to the lining. As soon as the lining is covered the "ring" increases more rapidly by building up upon itself around an ever-decreasing circumference until it seriously interferes with the operation, and has to be removed. It usually forms in the 15 or 20 ft. nearest the discharge end of the kiln, and puts an excessive amount of weight upon that end. A piece

of the "ring" is hard and has the appearance of having been partly fused. It looks very much like mill cinder.

Many devices have been tried to prevent the formation of the "ring," or to remove it without stopping the kiln, but, so far as I know, none of them have been successful. The best plan, in my experience, has been to shut down the kiln, when the ring became too large, cool it off with water, and cut it out with bars and picks. It is impossible, I feel sure, to prevent its formation when running on fine ores. With low-sulphur lump ores that do not need to be nodulized, careful control of the heat will allow a run of indefinite length, but as all the lump cinders (unless screened) carry a considerable percentage of fines (*i.e.*, −20 mesh), sooner or later a shutdown will be necessary. On any kind of cinder, however, the formation of the ring can be postponed by keeping, as nearly as possible, the conditions of temperature, speed, and feed, from fluctuating. The temperature should always be kept as low as is compatible with the successful operation of the kiln.

A 9-in. firebrick is generally used for lining, and is probably the best size on the larger kilns. A 4½-in. brick is satisfactory for kilns of 5 and 6 ft. diameter. The thicker bricks protect the shell better and prevent warping, but in the smaller kilns they reduce the available diameter too much. The lining wears down rapidly, and frequently individual bricks are loosened or broken in cutting out the ring. If the shell becomes warped under the tires, as is frequently the case when the kiln is not reinforced strongly enough at this point, it is very difficult to keep the bricks in. A 9½-in. lining should last over a year in the hot part of the kiln where the ring forms, and very much longer in the cooler end.

Automatic feeders are frequently used to insure a constant rate of charging, but they are really unnecessary, as the slow motion of the kiln will effect this, if only a simple feed pipe is used. All machinery used in handling the ore and the fuel and in rotating the kiln should be as nearly dust proof as possible, and should be driven by fully inclosed motors, as the fine cinder dust is very harmful. The best plan is to keep the feeding apparatus, the machinery for handling the roasted ore, and the fuel pulverizer and blower, all under separate housings. The kiln itself does not need to be housed, except where such housing is necessary to protect its driving machinery. Powdered coal is used as the fuel, so far as I know, exclusively, with the exception of one or two plants where blast-furnace gas has been tried. An oil flame would probably be very satisfactory where cheap oil could be obtained, on account of the ease of controlling the temperature incident to its use.

The cost of production of a ton of roasted ore, aside from the cost of the raw cinder, depends chiefly on the labor cost. Two men on a shift should be able to take care of a well-designed plant producing up to 100 tons in 24 hr., while in a poorly designed plant this number might easily

reach four or five. On a small kiln, 60 by 5 ft., the labor cost per ton, on high-sulphur fine cinder, averaged 40c. for several months, and with the same kiln, using high-grade lump cinder, this figure was reduced to 30c., but both these figures were excessive on account of faulty design, that made necessary three and four men on a shift; also on account of a very low output per man, due to the small size of the kiln. On the other hand, this was partly balanced by the fact that wages were low, and only two shifts were necessary. Repairs and supplies should not cost over 8c. per ton of roasted ore on a kiln of any size. The important factor in economic kiln operation is careful designing, and it cannot be said too strongly that the cost of operation of a rotary kiln will be almost entirely determined by the layout of the plant.

The roasted ore, for which the cement term of clinker is used, works very well in the furnace. In Virginia practice, where the blast pressure is comparatively low, it is not essential that the nodules should be large. If 80 per cent. of the clinker stays on a screen of 1/8-in. mesh, there will be no difficulty in the furnace at pressures of under 9 or 10 lb. It is very easy to make larger nodules, but as a higher temperature is required to do so, with the attendant evils of more frequent "ringing," greater fuel expense, and a harder, less porous product, it is better to keep the nodules as small as possible. The clinker frequently forms from one-sixteenth to one-half of the burden without ill effects, working with "brown ore" and specular hematites, and by its use the yield of the Southern furnaces where the native ores are lean can be raised from a scant 42 per cent. to 50 per cent. and over, depending on the quality of the cinder. It is in localities where the native ore is lean that the kiln is valuable. It affords an opportunity of enriching the burden with a high-grade ore at a much lower cost than could be done by direct purchase of lake or other rich ores, and where the silica in the native ore averages well over 20 per cent., it is indispensable that the burden should be sweetened in some way to cut down the excessive amount of flux and consequent fuel necessary to produce good iron.

Radiography of Metals

BY WHEELER P. DAVEY,* SCHENECTADY, N. Y.

(San Francisco Meeting, September, 1915)

In an article in the *General Electric Review*, January, 1915, reference was made to the X-ray examination of a steel casting 9/16 in. thick. Fig. 1 shows one of the radiographs thus obtained. All these radiographs showed plainly the tool marks on the surface of the casting. *All but one* showed peculiar markings the shape of which strongly suggested that they were indeed the pictures of holes in the interior. A circular piece, 1 in. in diameter, was punched from the casting at a point where the radiograph shown in Fig. 1 indicated that a blow-hole should be found. (Location of sample shown by circle in Fig. 1.) Fig. 2 is a photograph of the end of the punching, showing the hole that was found. Since that article was written it has seemed desirable: (1) to obtain data from which the exposure necessary for any thickness of steel could be at once calculated; (2) to find the thickness of the smallest air inclusion which could be radiographed in a given thickness of steel; (3) to determine in what direction to hope for further progress; and (4) to develop the technique of radiographing metals.

In order to gain some preliminary data, several pieces of ½-in. (12.5-mm.) boiler plate were obtained, 5 by 7 in. (12.5 by 17.5 cm.) in size. In one of these, holes were drilled in such a way that the axis of each hole was midway between the faces of the steel and parallel to those faces. The diameters of these holes were as follows:

Hole number	Diameter
1	¼ in. = 6.3 mm
2	⅛ in. = 3.1 mm.
3	1/16 in. = 1.6 mm.
4	1/32 in. = 0.8 mm.
5	1/64 in. = 0.4 mm.

Exposures were made on Seed X-ray plates at 20 in. (50 cm.) distance with Coolidge tube X-117, operated on Scheidel-Western induction coil with mercury turbine break. The X-ray plate was placed on a sheet of ⅛-in. (3.1-mm.) lead. The steel was placed above this and a lead

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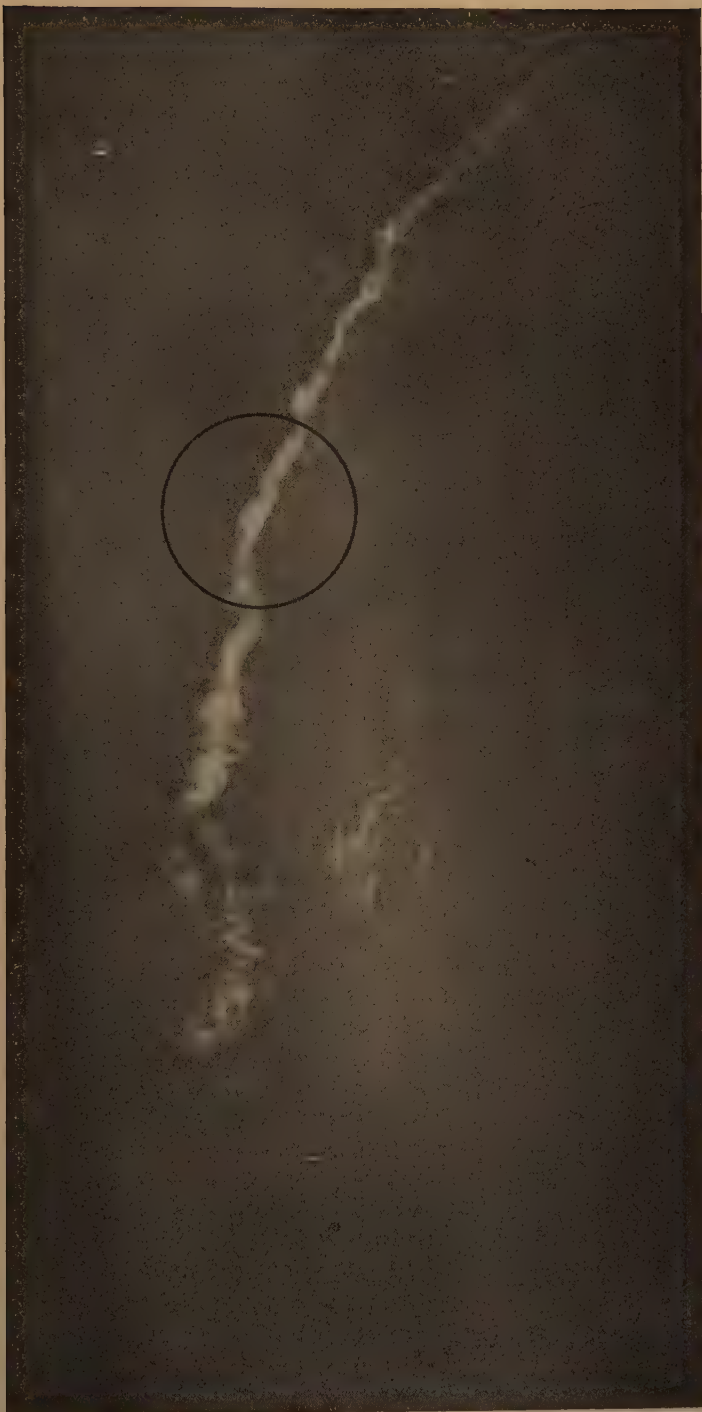

Fig. 1.—Radiograph of Steel Casting, $\frac{9}{16}$ in. Thick, Showing Flaw in Interior.

cover was placed over the whole in such a manner that the cover and backing made a complete lead shield for the X-ray plate. (See Fig. 3.) A rectangular hole in the cover allowed such X-rays as were able to penetrate the steel to reach the X-ray plate. This afforded complete protection

FIG. 2.—BUTTON, 1 IN. IN DIAMETER, CUT FROM STEEL CASTING.

against secondary rays. Without such precautions, the effect on the X-ray plate of secondary rays would have been greater than that of the rays used to take the picture. If the steel had been 2 or 3 ft. (60 or 90 cm.) square, such precautions would have been unnecessary. By placing the pieces of boiler plate on top of one another, any thickness of steel de-

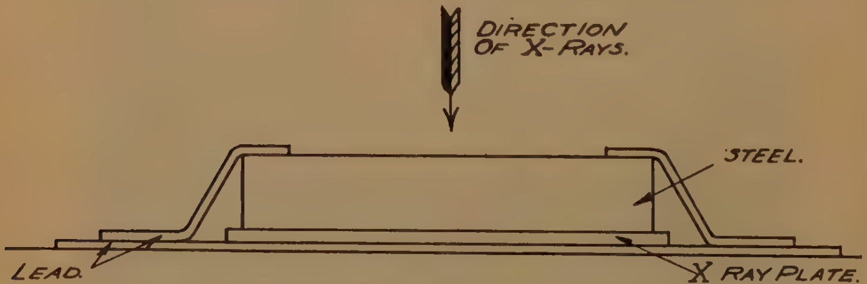

FIG. 3.

sired could be obtained. Exposures were made at 11, 13, and 15 in. (27.5, 32.5, 37.5 cm.) parallel spark gap between points. An attempt was made to use a 17-in. (42.5-cm.) spark gap, but was abandoned on account of flashing in the tube.

Results are tabulated below:

Thickness of Steel	Plate	Spark Gap	Exposure in MA-Min.	Holes Visible
		In. Cm.		
½ in.	D	11 27.5	7	1–2–3–4–5
(12.5 mm.)	A	13 32.5	4	1–2–3–4–5
	B	15 37.5	2	1–2–3–4–5
1 in.	E	11 27.5	45	1–2–3–4–5
(25 mm.)	F	13 32.5	19	1–2–3–4–5
	G	15 37.5	10	1–2–3–4–5
1½ in.	H	11 27.5	45	1–2–3 *very* faint
(37.5 mm.)	I	13 32.5	30	1–2–3 very faint
	K	13 32.5	90	1–2–3 faint
	J	15 37.5	30	1–2–3–4–5 very faint
	L	15 37.5	60	1–2–3–4–5 faint

This really means, of course, that at 13-in. (32.5-cm.) spark gap 90 milliampere-minutes is sufficient to enable one to notice the difference in blackening between exposures through 1$\frac{7}{16}$ and 1½ in. (36 and 37.5 mm.), but is not sufficient to enable one to detect the difference in blackening between exposures through 1$\frac{15}{32}$ and 1½ in. (36.8 and 37.5 mm.) of steel.

The above results were necessarily incomplete, since the plates were by no means all of the same density. They served, however, to show two things:

1. With the voltages which can now be used, it is impracticable to radiograph through more than 1½ in. (37.5 mm.) of steel with tungsten target tubes because of the time required.

2. The use of high voltages does not seem to reduce appreciably the clearness of the picture obtained. (It was to have been expected from the published data on scattering in aluminum that enough scattered radiation would have been produced to blur the pictures, but plate B apparently shows as good detail as does Plate D.)

It remained to confirm the above conclusions with data of a quantitative nature. Seed X-ray plates were therefore exposed under the same conditions as before, except that none of the slabs of steel used had been drilled. For each thickness of steel, all the exposures at a given spark gap were made on the same plate. Each plate, then, showed a series of steps which increased in density from one end of the plate to the other. Thickness of steel, spark gap, and milliampere-minutes were recorded on each plate by means of lead numbers. The following plates were thus exposed:

Plate	Thickness of Steel		Spark Gap	
No.	In.	Mm.	In.	Cm.
216	$\frac{1}{2}$	12.5	11	27.5
217	$\frac{1}{2}$	12.5	13	32.5
218	$\frac{1}{2}$	12.5	15	37.5
221	1	25.0	11	27.5
220	1	25.0	13	32.5
219	1	25.0	15	37.5
222	$1\frac{1}{2}$	37.5	15	37.5

A study of these plates showed the following facts:
Let $E_{\frac{1}{2}}$ be the exposure in milliampere-minutes necessary to produce a given darkening of the plate through $\frac{1}{2}$ in. (12.5 mm.) of steel, and let E_1 and $E_{1\frac{1}{2}}$ be the exposures necessary to produce the same darkening through 1 in. (25 mm.) and $1\frac{1}{2}$ in. (37.5 mm.) respectively.

Then at 11-in. (27.5 cm.) gap, $E_{\frac{1}{2}} : E_1 = 1 : 11$.

At 13-in. (32.5 cm.) gap, $E_{\frac{1}{2}} : E_1 = 1 : 8$.

At 15-in. (37.5 cm.) gap, $E_{\frac{1}{2}} : E_1 = E_1 : E_{1\frac{1}{2}} = 1 : 8$.

Also, through both $\frac{1}{2}$ in. (12.5 mm.) and 1 in. (25 mm.) of steel,

$$E_{\text{13-in. (32.5 cm.) gap}} : E_{\text{11-in. (28.5 cm.) gap}} = 1 : 4$$

and

$$E_{\text{15-in. (37.5 cm.) gap}} : E_{\text{13-in. (32.5 cm.) gap}} = 2 : 3.$$

It is at once evident that *either* X-rays from a tungsten target at 13-in. (32.5-cm.) gap are more penetrating than when produced at 11-in. (27.5-cm.) gap, *or* the X-ray plates used are more sensitive to the rays produced at 13-in. (32.5-cm.) gap. There is other evidence to show that the first of these conclusions is the more probable, but it is the *effect of the X-rays on the plate* which is of prime importance in this work, so that from a radiographic standpoint we may say in any case that the *effective penetration* of the rays is a little greater at 13-in. (32.5-cm.) than at 11-in. (27.5-cm.) gap.

In the same way we may conclude that the effective penetration at 15-in. (37.5-cm.) is the same as at 13-in. (32.5-cm.) gap. There is, however, a marked decrease in the amount of exposure required as the voltage across the tube (as measured by the spark gap) is increased. This may be due to one of two causes: *either* the efficiency of transformation from the kinetic energy of the cathode stream to the energy of the X-rays may be greater at the higher voltage, *or* there may be some peculiarity in the wave-form produced by the induction coil such that a great deal of energy is given off at a voltage corresponding to a 13-in. (32.5-cm.) gap when the coil is operated so as to give a maximum voltage corresponding to a 15-in. (37.5-cm.) gap.

From the data at hand it is easily possible by well-known means to construct formulas for computing the exposure necessary for radiographing steel at various spark gaps.

Let Q_0 be the quantity of X-rays impinging on the steel during the exposure.
Let Q be the quantity of the rays which pass through the steel.
Let x be the thickness of the steel.
Let λ be the coefficient of absorption, and
Let ϵ be the base of natural logarithms.

Then if the X-rays are homogeneous,

$$Q = Q_0\epsilon^{-\lambda x}$$

Now at 15-in. (37.5-cm.) gap we know that $\frac{E_{1/2}}{E_1} = \frac{E_1}{E_{1\frac{1}{2}}} = 1/8$.
The rays given off at 15-in. (37.5-cm.) gap are therefore practically homogeneous. Since $\frac{E_{1/2}}{E_1} = 1/8$ at 13-in. (32.5-cm.) gap, we may assume that these rays are also practically homogeneous. Rays given off at 11-in. (27.5-cm.) gap are still sufficiently homogeneous, after having passed through the first few hundredths of an inch of steel, to permit their being treated as though they were actually homogeneous. Calculations for exposures at 11-in. (27.5-cm.) gap are to be considered as only good approximations.

For 15-in. (37.5-cm.) gap we have

$$Q/Q_0 = 1/8 = \epsilon^{-\lambda x} = \epsilon^{-\frac{1}{2}\lambda}$$
$$\log 8 = \tfrac{1}{2}\lambda = 2.079$$
$$\lambda = 4.16 \text{ in.}^{-1} = 1.64 \text{ cm.}^{-1}$$

Likewise for 13-in. (32.5-cm.) gap

$$\lambda = 4.16 \text{ in.}^{-1} = 1.64 \text{ cm.}^{-1}$$

Applying the same method for 11-in. (27.5-cm.) gap

$$\lambda = 4.80 \text{ in.}^{-1} = 1.89 \text{ cm.}^{-1}.$$

Now at 15-in. (37.5-cm.) gap and 20 in. (50 cm.) distance 0.8 milliampere-minutes gives a good exposure through ½ in. (12.5 mm.) of steel. A corresponding darkening would have been produced on a bare plate by an exposure of 0.1 milliampere-minutes. This corresponds to Q in our formula. We may therefore write, since $Q_0 = E$,

$$0.1 = E \, . \, \epsilon^{-4.16x}$$
$$10E = \epsilon^{4.16x}$$
$$\log_\epsilon 10E = 4.16x$$
$$\log_{10} 10E = 1.80x$$
$$E = 1/10 \log_{10}{}^{-1}. \, 1.80x.$$

where x is the thickness of the steel in inches

or
$$E = \tfrac{1}{10} \log_{10}^{-1} 0.71x$$

where x is the thickness of the steel in centimeters.

The corresponding formulas for 13-in. (32.5-cm.) gap are

$$E = \tfrac{3}{20} \log_{10}^{-1} 1.80x \quad (x \text{ in inches})$$
$$E = \tfrac{3}{20} \log_{10}^{-1} 0.71x \quad (x \text{ in centimeters}).$$

FIG. 4A.

FIG. 4B.

The approximate formulas for 11-in. (27.5-cm.) gap are

$$E = \tfrac{3}{5} \log_{10}^{-1} 2.09x \quad (x \text{ in inches})$$
$$E = \tfrac{3}{5} \log_{10}^{-1} 0.82x \quad (x \text{ in centimeters}).$$

It remained to find the thickness of the smallest air-inclusion which could be radiographed in steel at 15-in. gap. For this purpose two plates

of steel were taken. The faces were machined flat and in one of them a slot was cut, thus giving a *wedge* of air. The slot and the faces of the steel plates were then ground smooth. When completed, each plate was $5/8$ in. (15.5 mm.) thick. The air wedge was 10 in. (250 mm.) long, 1 in. (25 mm.) wide and $9/64$ in. (3.52 mm.) thick at its thick end. When the two plates were bolted together, the air wedge simulated a blow-hole in a

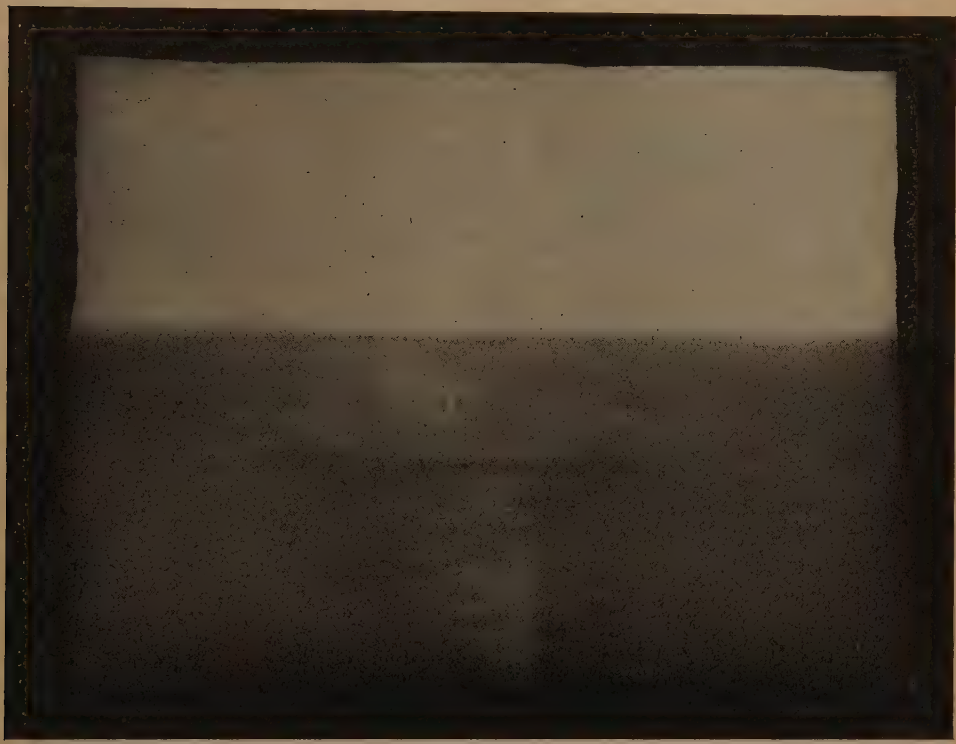

Fig. 5a.

Fig. 5b.

casting. The wedge was then radiographed at 15-in. (37.5-cm.) gap. When the X-ray plates were dry the place was noted at which the outline of the wedge was barely visible. In order to avoid error, only a small portion of the wedge was viewed at one time, the rest being blocked off with cardboard. It was found that an air inclusion 0.021 in. (0.52 mm.) thick could be detected in $1\frac{1}{4}$ in. (31.3 mm.) of steel. In $5/8$ in. (15.6 mm.), an air inclusion of 0.007 in. (0.18 mm.) could be detected.

Besides the work above outlined, a great deal of work has been done in the actual taking of pictures, so that the technique of radiography through metals might be worked out.

A record of a single example will suffice. Four samples of autogenous welds in steel were obtained. The welding had been done with an oxy-acetylene flame. The samples were ½ in. (12.5 mm.) thick and

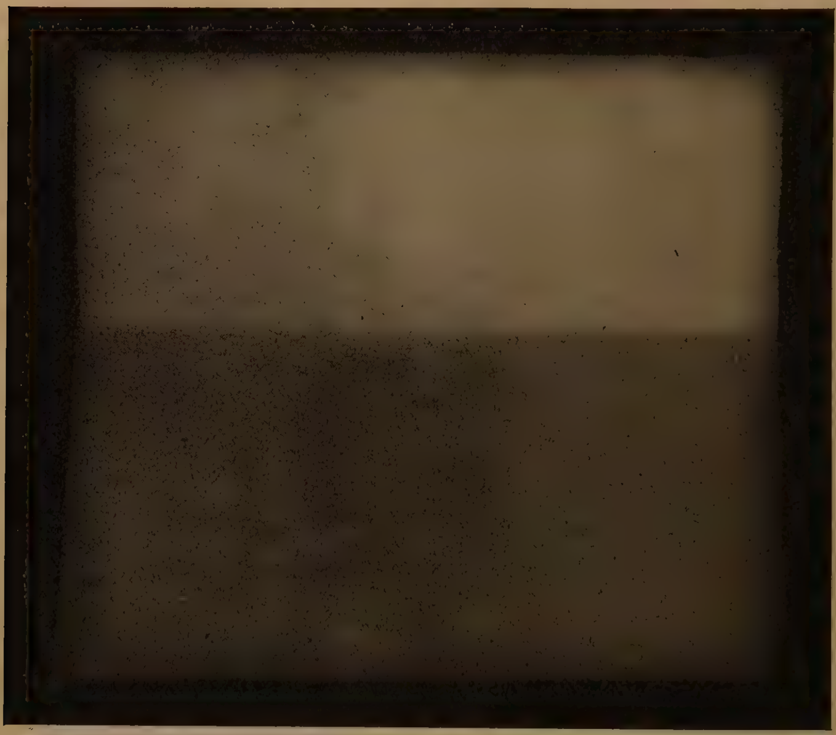

Fig. 6A.

Fig. 6B.

about 4 in. (10 cm.) square. The faces were fairly rough. Sample No. 1 had only been welded on the surfaces. (See Fig. 4B.) Sample No. 2 had been insufficiently heated so that there was incomplete fusion of the metal at the center. (See Fig. 5B.) In welding sample No. 3 an excess of oxygen had been used in the flame, causing the presence of oxide on the surface. (See Fig. 6B.) Sample No. 4 was considered to be a good weld. (See Fig. 7B.) One-half of each face was machined off, so that half the

length of the weld was between flat, parallel faces; the other half was left under the original rough surfaces. As a result, one-half of each sample was $\frac{1}{2}$ in. (12.5 mm.) thick and the other half was about $\frac{3}{8}$ in. (9.4 mm.) thick. Radiographs were taken at 15-in. (37.5-cm.) gap under the conditions described above. Reference to the formula for exposure at 15-in. (37.5-cm.) gap shows that the exposures through the $\frac{1}{2}$-in. (12.5-mm.) and

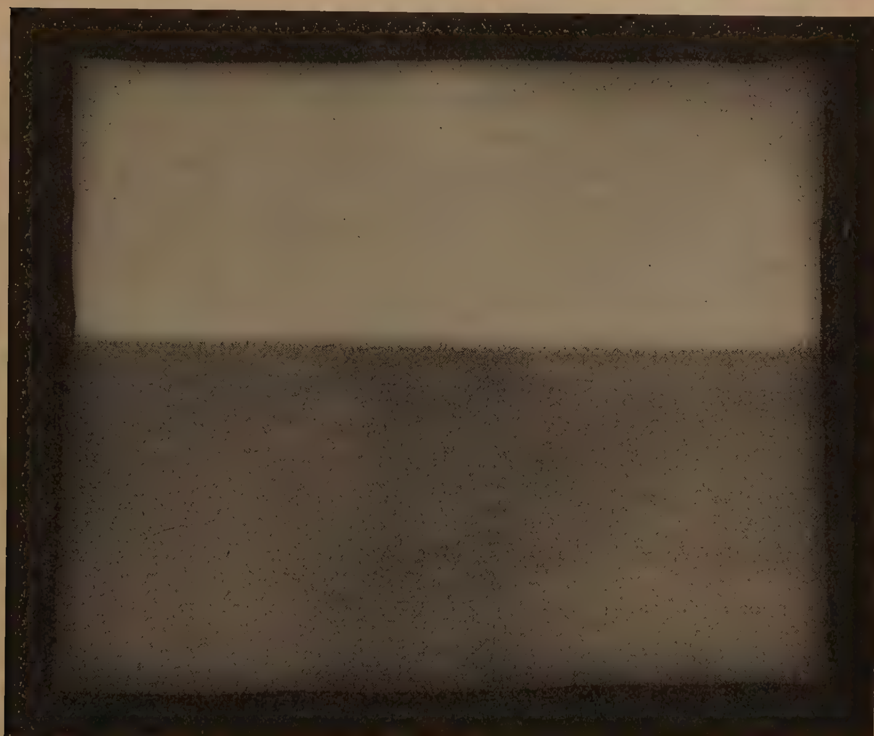

FIG. 7A.

FIG. 7B.

$\frac{3}{8}$-in. (9.4-mm.) portions were in the ratio of 1 to 1.7. The resulting radiographs are shown in Figs. 4A, 5A, 6A, and 7A.

Fig. 4A shows clearly the unwelded center of sample 1 in both portions of the picture. Fig. 5A shows, in both portions of sample 2, the holes caused by the metal not having been thoroughly fused at the center. That portion of Fig. 6A which was taken through the machined end of the weld of sample 3 would seem to indicate a porous structure. Such a

structure was evident during the machining. The portion of the picture taken through the unmachined end of the weld did not show such a structure with certainty. This was to have been expected, as the inequalities in thickness due to the uneven surface were at least as great as those due to porous or frothy structure. Fig. 7A shows that as far as gross structure is concerned, sample 4 was a good weld.

It is of course self-evident that a radiograph gives only the gross structure of the metal, and no information as to the "grain," crystal interlocking at the edge of the weld, etc. A radiograph does, however, give valuable information as to the presence of blowholes, slag inclusions, porous spots and defects of like nature, which could not be found otherwise except by cutting into the metal. Unfortunately, no fluoroscopic screen now known is sensitive enough for this work; all work in metals must be done radiographically. An inspection of the formulas derived above makes one feel that, for the present at least, radiography of steel is a commercial possibility only up to thicknesses of ½ in. (12.5 mm.). For greater thicknesses, the time required is excessive. The great saving in time which is gained by the use of a 15-in. (37.5-cm.) instead of a 13-in. (32.5-cm.) spark gap makes it seem probable that a further increase in the voltage across the tube would allow us to radiograph still greater thicknesses of steel.

Suggestions Regarding the Determination of the Properties of Steel

BY ALEXANDRE MITINSKY,* PETROGRAD, RUSSIA

EDITED BY LAWFORD H. FRY

(San Francisco Meeting, September, 1915)

THE theory of elasticity, the science of the strength of materials, and all our calculations regarding engineering structures are based on Hooke's law, that in loaded bodies the deformations are proportional to the load producing them. Without this law it would be impossible to design intelligently engineering structures such as bridges, frames, axles, etc. This being the case, the proportional limit,[1] that is, the point at which deformation ceases to be proportional to load, is the most important factor in engineering design. Further, the working fiber stress should be fixed in reference to the proportional limit instead of, as is usual, in reference to the ultimate strength. The incorrect use of the ultimate strength instead of the proportional limit in setting the working fiber stress is probably due to two reasons: (1) it is a more delicate operation to measure the proportional limit; (2) it is usually assumed, though incorrectly, that the proportional limit is determined by the ultimate strength.

In Germany the point at which permanent set first appears is often taken instead of the proportional limit. The two points will coincide if Gerstner's law, that elastic deformations are proportional to the loads producing them both before and after permanent set, is true; and the limit of proportionality and the elastic limit or point of permanent set will both be represented on a stress-strain diagram, by the point at which the line ceases to be a straight line and becomes curved or changes its inclination. It is to be further noted that the difference in definition between proportional limit and elastic limit introduces a difference in methods of determination in the testing laboratory. The German method of determining the elastic limit by alternately loading and unloading the test specimen is a lengthy process and time has in itself a very considerable influence on the results of physical tests.

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[1] EDITOR'S NOTE.—In the MS., Mr. Mitinsky speaks throughout of the "elastic limit," but he defines this as "the point at which the extensions cease to be proportional to the loads." Therefore according to American terminology he is referring to the "proportional limit" and this term has been used throughout instead of "elastic limit" as originally written.—L. H. F.

In this way a difference in definition may lead to a difference in the determination of the point sought. The real danger from erroneous ideas regarding the material to be tested lies, however, not in this possible difference of measurement, but in the mistake, so common in Europe, of considering the yield point as the "practical elastic limit." This is only true in the sense that the yield point can be measured practically and simply, but is not true in the sense that a measurement of the yield point will give a correct idea of the elastic qualities of the metal tested. The value of the yield point has no relation to the elastic properties of the material. If the yield point stood in fairly close connection with the proportional limit we could use it in judging of the quality of the metal, but unfortunately this is not the case.

The stress-strain diagrams of steel can be divided into two groups, of which Figs. 1 and 2 are typical.

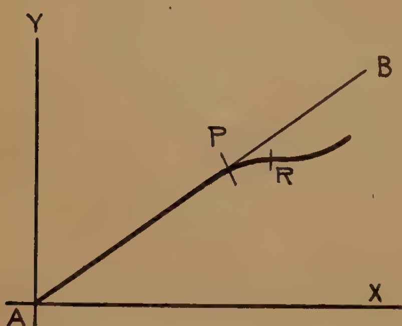

FIG. 1.

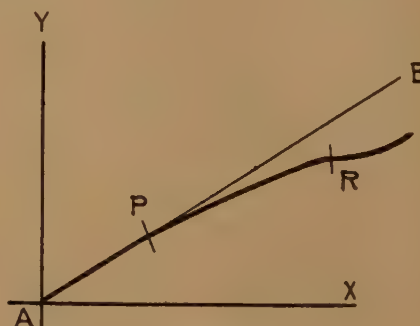

FIG. 2.

In these figures AB is the line representing proportionality between stress and strain, P is the proportional limit, and R the yield point.

Fig. 1 corresponds to a good steel which has been correctly treated both mechanically and thermally. This steel has the proportional limit and the yield point close together.

Fig. 2 represents a steel of poor quality. The proportional limit P is low, and beyond this point, P, the stress-strain line departs but little from the proportional line AB, until the yield point, R, is reached. The steel with a diagram of this character may show good elongation and reduction of area and a high ultimate strength, but is extremely undesirable for railroad service.

From a physical point of view, the proportional limit is the load at which the shearing stresses produced by a tension or compression load become sufficiently large to break down the metal. At the moment when this occurs, Hartmann's lines appear on the surface of the test specimen, which becomes dulled, and at the same time the temperature

rises. After a stress of this magnitude has been applied the metal is not in the same condition as before, its internal structure having been deformed.

By gradually loading and unloading a test specimen the proportional limit can be raised, and in many cases where the stress-strain diagram is of the type shown in Fig. 2, the increase is a substantial one. The proportional limit cannot, however, be carried above the yield point, and any blow or shock will reduce the artificially raised limit. It is further to be noted that this artificial raising of the proportional limit in tension is accompanied by a corresponding reduction in the proportional limit in compression, and inversely.

By subjecting metal to repeated and alternating tension and compression metal can be destroyed by so-called fatigue by loads very much below the ultimate strength. The resistance to fatigue depends solely on the proportional limit of the metal and is not due to some other special property. If the stress under repeated loads is less than the proportional limit no change occurs in the metal and no repetition of such a stress will cause injury, since the internal structure of the metal remains undisturbed. On the other hand, if the repeated stress exceeds the proportional limit the metal will be deformed and eventually ruptured. On the first application of the load producing such a stress, say in tension, a permanent deformation is produced, the shearing stresses do internal work, and the elastic limit in tension is increased. When the load is reversed on the following alternation the previously extended fibers are compressed and permanent deformation in compression is produced. This is the greater because the elastic limit in compression has been lowered by the previous deformation under tension. This process repeats itself, the metal being increasingly changed internally at each alternation.

In so far as fatigue of metal is concerned the proportional limit may be compared to the critical temperature in physics.

It would appear from the foregoing that the fatigue test, with stresses above the proportional limit, which is required by some English railways for tires and axles, could be replaced in practice by a determination of the proportional limit. It is sometimes said that a determinaton of the proportional limit is an impossibility in ordinary commercial testing, and doubtless the usual German method of making such determinations by applying and releasing the load has done much to justify this opinion. It is, however, a fact that in the Russian railway material inspection tests, determinations of the proportional limit are currently made and do not occupy more than 15 to 20 min. each. Marten's mirror apparatus and the Ewing and the Cambridge Scientific Co.'s extensometers are used. In this inspection testing it is not of importance to determine the exact value of the proportional limit, but rather to make sure that it is not less than 20 or 25 kg. per square millimeter (28,400 or 35,500 lb. per

square inch), as the case may be. In case the proportional limit is found to be less than the specified value the tires represented are rejected.

Having established the possibility of using the proportional limit in the specification, the Russian railways are now introducing it into their rail specifications, the feeling being that it is preferable to reduce the numbers of tests if necessary provided that the tests have a real significance.

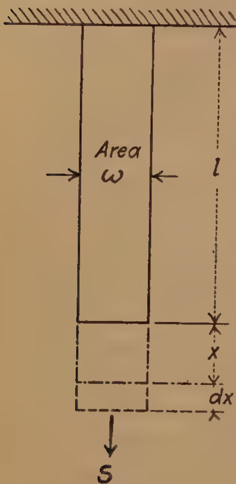

FIG 3.

If a bar of unstrained length l, Fig. 3, has a load gradually applied, the work absorbed while the elongation is increased from x to $x + dx$ is Sdx, where S is the amount of load corresponding to the elongation x.

Writing W for the work absorbed by the deformation of the bar we have the equation

$$W = Sdx \tag{1}$$

Now if w be the area of the bar, v its volume, so that $v = wl$, and E the modulus of elasticity

$$\frac{S}{w} = \frac{Ex}{l}$$

so that

$$W_1 = \frac{Ew}{l} xdx \tag{2}$$

If this be integrated for x between the limits 0 and λ, we get

$$W = \frac{Ew}{2l} \lambda^2 \tag{3}$$

where W is the work absorbed, or the internal work in the bar during its extension to the length $l + \lambda$. Again, if S be the load and Q the fiber stress corresponding to the extension λ

$$Q = \frac{S}{w} = \frac{E\lambda}{l}$$

and consequently

$$W = \frac{Q^2}{2E} v \tag{4}$$

Hence $\frac{Q^2}{2E}$ measures the internal work absorbed by the bar per unit of volume while being loaded to the stress Q. It therefore follows that for material with a proportional limit P the resistance to shocks and external work of any kind produced by loading is measured by $\frac{P^2}{2E}$, and that if a greater amount of work per unit of volume is to be absorbed a permanent deformation will take place. The modulus of elasticity being practically

the same for steels of the same analysis though differently treated (burning excepted), the resistance of the steel to blows, etc., is proportional to the square of the value of its proportional limit. For example, two pieces of tire steel both having 65 kg. per square millimeter (92,500 lb. per square inch) ultimate may have proportional limits of respectively 10 and 25 kg. per square millimeter (14,200 and 35,500 lb. per square inch) if the one is poorly and the other properly annealed. In this case the resistances of the two tires would be in the proportion of 1 to 6, while if it be remembered that the shrinkage which holds the tire on the wheel sets up an initial stress of say 8 kg. per square millimeter (11,300 lb. per square inch) the difference will be found to be still greater, say 1 to 70.

Ewing has proved that the elongation (l) or the compression produced by the shock of two bodies depends, not on the mass or dimensions of the bodies, but only on the relation between the velocity (v) of the body and on (c) the velocity of sound in the material under consideration; in other words, $l = \frac{v}{c}\cdot$ Now the metal will be deformed if l is sufficient to produce a stress equal to P, the proportional limit; then under these conditions, with unit length $P = lE$, and combining this with the preceding equation, we have as a value for v_m the velocity of shock which will stress the body to the proportional limit

$$v_m = \frac{cP}{E} \tag{5}$$

Now in steel the velocity of sound (c) is about 4,900 m. (16,000 ft.) per second, and E = 22,500 kg. per square millimeter (32,000,000 lb. per square inch), from which it follows that to 10 kg. per square millimeter (14,200 lb. per square inch) of proportional limit there corresponds a dangerous velocity of shock of about 7 km. (4.4 miles) per hour. A good rail having a proportional limit of 25 to 30 kg. per square millimeter (35,500 to 42,700 lb. per square inch) will therefore have a dangerous velocity of shock of 18 to 21 km. (11 to 13 miles) per hour. According to Boussinesq the dangerous velocity for bending stresses is double that for tension. It is possible that the relation is not exactly as outlined above and that some coefficient should be introduced into equation (5), but it is evident that a rail or a tire with a high proportional limit is more likely to give satisfactory service than would one with a low value for this property.

The wear of rails and tires is of two kinds, (1) a crushing action producing a flow of the metal, and (2) an abrasive action. Wear of the first kind, or wear by deformation, depends, as has been just explained, only on the value of the proportional limit. Practical experience has shown that failure by deformation may occur in rails with high as well as with low tensile strength, but that it is in every case accompanied by a

low value of the proportional limit and that it is independent of the value of the yield point.

In the years from 1899 to 1906 a special committee was appointed by the Russian government to investigate the causes of rail wear and to work out means for securing satisfactory service from rails. This committee studied the question very thoroughly. A large number of rails made by manufacturers in Russia and other countries were subjected to chemical analysis and numerous physical tests, the rails being chosen so as to include those which had given the best as well as those having given the worst results in service. Table I sums up some of the results obtained:

TABLE I

Millions of Tons Carried by Rail	Number of Rails	Per Cent. of Carbon			Ultimate Tensile Strength, Pounds per Square Inch			Brinell Hardness Number		
		Min.	Max.	Mean	Min.	Max.	Mean	Min.	Max.	Mean
10 or less	46	0.165	0.615	0.411	72,500	118,000	92,500	163	233	194
10 to 50	72	0.127	0.705	0.342	65,400	118,000	88,200	144	246	191
Over 50	19	0.135	0.725	0.375	68,300	102,400	86,800	139	224	192

All tests for Russian railways are made using the International standard test specimen and the metric system.

In addition to other tests, drop tests were made on pieces of rail 5 ft. long, by dropping a ½-ton tup from a height h determined by the formula

$$h = 0.25\frac{I}{z^2} \tag{6}$$

where I is the moment of inertia of the section and z the distance of the most strained fiber from the neutral axis. The permanent deflection after the first blow was noted in each case, the figures obtained being as follows: For good rails having given long service from 1.8 to 3.2 in., average 2.5 in., the best showing 2.6 in.; for rails with crushed heads 2.2 to 3.0 in., average 2.6 in.; for rails rapidly worn by abrasion 2.1 to 3.1 in., average 2.5 in. The committee was unable to establish any definite relation between the wear obtained from the rails in service and the ultimate strength, Brinell hardness, yield point, elongation, reduction of area, chemical analysis, or deflection on drop test. As a consequence little attention is now paid in Russia to ultimate strength and chemical analysis.

About four years ago over 20,000 tons of rails on the Siberian Railroad were found to be defective. These rails, which were all from the same steel works, showed crushed heads soon after having been laid. Table II herewith summarizes the results obtained by the Laboratory of Means of Communication at Petrograd. The first 12 lines give particulars ob-

tained from rails which had been in service, while the last seven lines refer to rails selected at the works. Several test pieces were taken from each rail and the maximum and minimum results are given in the table. It is evident that all 12 rails which crushed in service had low values for the proportional limit. In some cases it was only one-quarter the yield point. On further investigation it was found that the works, in the endeavor to secure a large output of rails from a weak mill, had finished them too hot.

TABLE II

C	Mn	Si	P	S	Ultimate Strength Kg. per Sq. Mm.	Elongation Per Cent.	Reduction of Area Per Cent.	Yield Point Kg. per Sq. Mm.	Proportional Limit Kg. per Sq. Mm.	Brinell Number
0.55	0.35	0.011	0.036	0.010	65 to 67	11.5 to 17	27 to 35	38 to 45	9.5 to 9.5	158
0.55	0.22	0.007	0.069	0.006	68 to 73	13.5 to 19	27 to 37	27 to 39	9.5 to 10.5	161 to 174
0.45	0.44	0.034	0.043	0.007	62 to 64	13 to 14	44 to 46	38	16	150
0.63	0.39	0.013	0.037	0.005	71 to 71	14	24 to 28	28	9.5	180
0.54	0.42	0.012	0.055	0.029	63 to 66	15 to 19	31 to 42	40 to 47	10 to 14.5	156 to 167
0.37	0.22	0.009	0.051	0.020	58 to 61	11 to 20	44 to 50	35 to 46	16	154 to 158
0.38	0.44	0.012	0.085	0.019	59 to 63	13 to 44	43 to 50	43	16	158
0.48	0.66	0.018	0.035	0.025	66 to 68	13 to 19	37 to 40	45 to 50	14.5 to 16	158 to 167
0.42	0.86	0.040	0.087	0.008	61 to 68	11 to 20	51 to 55	41 to 55	16 to 22	156 to 176
0.56	0.59	0.012	0.046	.....	78 to 80	8 to 15	11 to 24	45 to 47	13 to 16	187 to 194
0.48	0.57	0.012	0.031	0.018	65 to 70	13 to 18	26 to 40	42 to 49	10 to 11	160 to 176
0,54	0.35	0.018	0.036	0.023	63 to 68	19 to 20	35 to 40	35	9.5	153
0.69	0.44	0.028	0.050	0.029	85 to 87	9	16	49	12.5	194
0.65	0.73	0.170	0.083	0.021	73	17 to 18	31 to 36	41	19.5	168
0.61	0.18	0.090	0.045	0.029	75 to 76	14	21 to 25	41	20.5	174
0.50	0.36	0.030	0.035	0.021	68 to 69	17 to 18	29 to 36	45	16.0	163
0.63	0.25	0.017	0.053	0.021	79 to 80	13 to 14	26 to 27	47	20.5	185
0.55	0.39	0.090	0.043	0.038	73 to 75	15 to 16	22 to 32	34	16.5 to 24.5	174 to 180
0.57	0.36	0.110	0.060	0.039	80 to 81	11 to 14	21 to 25	37	29.0	194

The investigation of the influence of the proportional limit was continued by an examination of 12 tires from locomotives in main-line service on one of the Russian railroads. The tires were chosen from those removed from service after having reached the normal limit of wear, those showing the best mileage being selected. Three test pieces were cut from each, as near to the outer circumference as possible: Sample I from the flange, II from the center of the tread, and III from the face edge of the tread. The results are shown in Table III.

All of these tires which have given good results in service show a high value for the proportional limit, the average for all being 39,000 lb. per square inch.

If the rolling of a rail is finished at a high temperature the value of the proportional limit will be low, so that a measurement of the proportional limit will show whether or not the rolling was properly done. Again, tires will show a low proportional limit if improperly annealed; that is, if the critical temperature of say 875° C. has not been reached, or if the internal critical temperature has not been passed rapidly enough

Table III

Total run in thousands of versts[a]	363	337	337	329	289	284	273	238	171	165	164	152
Years of service	11¾	12 1/12	10¼	11¼	12 1/6	13 1/6	7 5/6	9 11/12	6 7/12	10 1/6	6 1/12	5 5/12
Load per axle, tons	13.3	13.3	13.3	13.3	13.3	13.3	14.4	14	12.8	14.4	14.4	13 3
Ultimate strength, kg. per sq. mm.:												
I	70	71	69	64.5	70.5	70.5	67.5	......	......	75.5	57.500	69.5
II	63.5	74	67	62	67.5	70	69.5	73	69	78	52.500	65.5
III	68.5	74	70	67	68.5	71	71.5	77	64.5	83	55.000	76.0
Elongation, per cent.:												
I	8	6.5	14	23.5	20	15.5	14.5	......	......	13	21.000	14.5
II	17.5	13	18	17	15.5	16.5	15	17	15	12.5	28.500	13.5
III	20	13.5	14.5	11.5	6.5	15.5	10.5	12	13.5	13.5	23.500	12.5
Reduction of area per cent.:												
I	......	13	31	31	40	39	28	......	......	20	34.000	35.0
II	62	29	39	41	24	32	34	33	39	23	43.000	46.0
III	39	23	38	26	10	35	32	28	42	30	45.000	23.0
Proportional limit, kg. per sq. mm.:												
I	29	27	26	23	33	28.5	18.5	......	......	21	22.000	29.50
II	22.5	25.5	19	27	30	31.5	29.5	28.5	27.5	32	17.000	31.50
III	31.5	25	26	35.5	32	25	22.5	31	27	40	20.500	25.50
Average	28	26	24	28.5	32	28	27	30	27	31	20.000	29.00
Carbon	0.460	.....	....	....	....	....	0.500	0.560	0.470	.....	0.420	0.460
Manganese	0.980	.....	....	....	....	....	1.020	1.100	1.060	.....	0.780	1.070
Silicon	0.090	.....	....	....	....	....	0.121	0.197	0.181	.....	0.191	0.292
Phosphorus	0.026	.....	....	....	....	....	0.036	0.021	0.025	.....	0.027	0.039
Sulphur	0.018	.....	....	....	....	....	0.022	0.033	0.018	.....	0.044	0.020

[a] One verst is equal to 1.065 km.

during cooling. An unannealed tire rolled at too high a temperature will also show a low value for the proportional limit.

Generally speaking, a steel with a medium amount of carbon and having a fine regular internal structure will have a high value for the proportional limit. There are, however, two cases in which a high proportional limit does not correspond to a good steel. These are: (1) overheated steel, Widmanstedt's structure; (2) high phosphorus or arsenic, or too much manganese in combination with a comparatively high percentage of carbon. In both cases the metal with a high proportional limit may be brittle, but this condition will be disclosed by a drop test.

It may therefore be concluded that if a rail or a tire shows a high proportional limit and sustains satisfactorily a drop test, the steel is of as good a quality as can be desired, and no attention need be paid to the ultimate strength or Brinell hardness (these two are practically the same thing), nor to the elongation, reduction of area, carbon, manganese, etc.

The foregoing has dealt with wear of rails and tires by flow of metal. The wear by abrasion has not yet been studied as closely as it deserves. Practically all that can be said at present is that sulphur in steel tends to increase abrasion as it segregates sharply and may be present on the wearing surfaces in considerably higher quantities than in the average analysis of the ingot. Many persons are of the opinion that the ultimate strength has a great influence on the resistance of the steel to abrasion, but the author has been unable to find any sound basis for this opinion.

At the meeting of the International Society for Testing Materials in New York in 1912, Robin, Saniter, Rosenhain, and at Copenhagen in 1908, Nusbaumer proved that this opinion is erroneous, and the same conclusions were reached by the Russian Rail Committee referred to above. The author therefore feels that the only honest answer to the question "On what does the abrasive wear of steel depend?" is "Except in the particular case of sulphur, we do not know."

It is sometimes said that the Brinell hardness number bears a relation to the abrasion, but the absence of such a relation was shown by the same tests as disproved the theory of a relation between abrasion and ultimate strength. The reason is clear. The scientific determination of hardness by Hertz in 1882, defines it as the value of the normal pressure per unit of area at the center of the bearing surface of a spherical body at the moment in which the elastic limit is attained. Auerbach was the first to carry practical tests beyond the elastic limit, and in all Brinell's experiments this limit is exceeded and permanent plastic deformations alone measured. It follows that Brinell's hardness numbers can be related only to the ultimate strength and are quite foreign to the true hardness and therefore to the wear by abrasion.

The author formulates the following suggestions regarding the quality of metal generally:

1. The proportional limit is the proper basis of all engineering calculations.
2. The proportional limit is independent of the yield point and of the tensile strength.
3. The proportional limit is the most important property, and should be as high as possible, especially for metals subjected to repeated loads, shocks, wear by crushing, etc.
4. The wear of metals of all kinds is independent of the ultimate strength, yield point, and Brinell number.
5. Wear by flow of metal or crushing depends only on the proportional limit. Wear by abrasion is not completely studied and all that can be said is that sulphur has a deleterious influence.
6. The proportional limit can and should be measured in the engineering inspection of materials.
7. The allowable working stress of any material should be fixed as a proportion of the proportional limit.
8. The resistance to fatigue is directly dependent on the proportional limit, and provided this is not exceeded no number of repeated stresses will destroy the metal.

Since this paper was written the author has received Mr. Strohmeyer's report, and is now quite sure that his fatigue limit corresponds exactly to the proportional limit.

Discussion

Lawford H. Fry, Burnham, Pa. (communication to the Secretary*). —The author assumes that the proportional limit and the elastic limit are coincident or very nearly so, and then further assumes that the stress corresponding to these two points is the least stress which if repeated will cause failure. The point is extremely interesting and it is to be regretted that the author did not quote experimental data to show the relation between the limit of repeated stress and the other properties of the material.

The author's suggestion that a steel of poor quality may have a normal value for the yield point and tensile strength (see Fig. 2) and an abnormally low elastic limit, and that this will correspond to a low resistance to repeated stress, may account for a number of failures of axles in which the tensile tests show normal physical properties. If so and if a determination of the proportional limit would have eliminated the material before it was put into service, this determination will merit most earnest attention. It appears, however, that considerable further research is necessary before any definite conclusion as to the quality of a steel can be drawn from its proportional limit. Mr. Mitinsky's paper is extremely interesting and if it stimulates thought and work along the lines indicated, it may prove of considerable practical benefit.

H. V. Wille, Philadelphia, Pa.—The paper discusses the difference between the proportional limit, or elastic limit, and the yield point, and refers to the fact that much harm results from confusion of the terms yield point and elastic limit. Similar confusion has arisen in this country; the American Society for Testing Materials does not even consider the elastic limit and proportional limit to be identical and recently adopted the following definitions:

Elastic limit is the least load per square inch which produces a permanent set as indicated by an extensometer with a degree of precision of 0.0001 in.

Proportional limit is the load per square inch where stress (load per square inch) and deformation (elongation) per unit of length cease to be proportional to each other, the deformation being determined by an extensometer with a degree of precision of 0.0001 in.

Yield point is the load per square inch at which a distinctly visible increase occurs in the distance between the gage marks on the test specimen, as observed by using accurate dividers or an equivalent instrument, or at which, when the load is increased at the rate hereafter specified, there is a distinct drop of the beam on the testing machine.

It is customary in this country to determine the yield point for the

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*Received June 3, 1915.

softer grades of material, but the specifications distinctly require that the yield point is to be determined so that there can be no error resulting from the incorrect interpretation of the results.

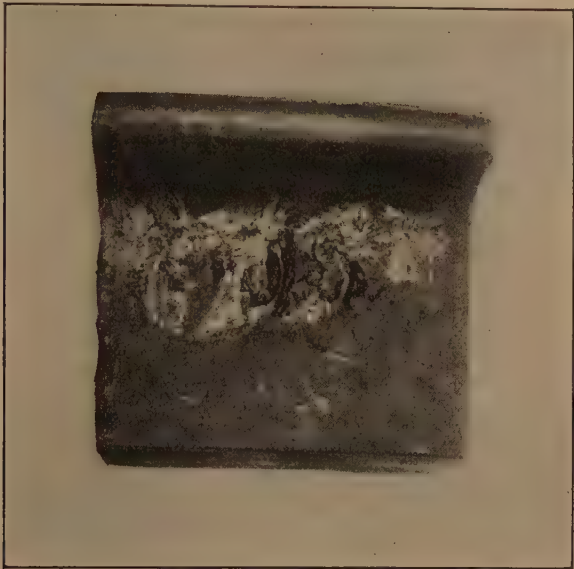

FIG. 1.—PORTION OF TIRE WHICH HAS DEVELOPED A SHELL SPOT IN SERVICE.

Specifications for heat-treated material usually require a determination of the elastic limit and so that there can be no misunderstanding it is required to determine the elastic limit by means of extensometers.

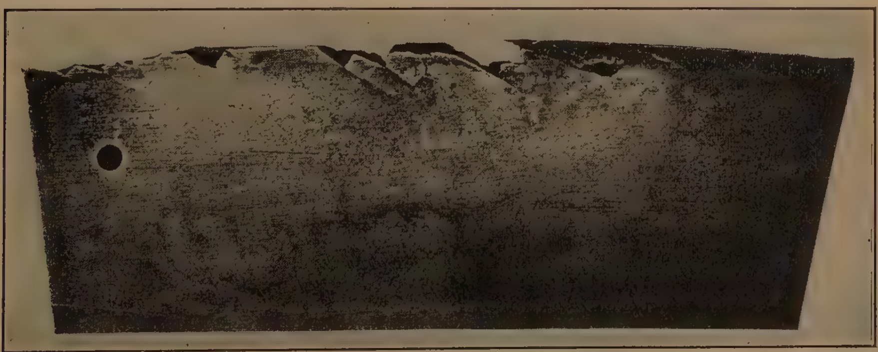

FIG. 2.—RADIAL SECTION OF TIRE, SHOWN IN FIG. 1, HEATED AND QUENCHED IN WATER.

The determination of the elastic limit, of course, requires the removal of the load after every measurement is taken, in order to determine

any permanent set in the test piece. The method followed is essentially the same as employed in the German laboratory. The proportional limit, however, can be determined without removal of the load, and can, therefore, be more readily determined than the elastic limit.

I have often observed a point of deflection in the stress-strain diagram which would indicate that the material had a very low proportional limit, but we have usually found this point of deflection to be the result of some inaccuracy in the measurements made by the extensometer as a result of one side of the test piece stretching more than the other side. None of our experiments show that it is possible to secure an abnormally low proportional limit in a bar which has been annealed at a rather high temperature or in one which has been finished at a high temperature. If the material has been manufactured from a homogeneous ingot our results show that the proportional limit is never less than the values obtained upon a thoroughly annealed sample of material. These results seem to be confirmed by those of Mr. Bauchinger who maintains that the number of reversals of reversed stresses of a given grade of material depends upon its homogeneity. Our own experiments on tires bear out this statement.

Fig. 1 shows a portion of a tire which developed a shell spot in service.

Fig. 2 shows a radial section of this tire which has been heated and quenched in water to develop any cracks which existed in the section of the tire. It will be noted that the tire contains a number of cracks running at an angle of 45° through the periphery of the tire. These cracks are precisely of the same nature as those developed in testing a cylinder in compression. The cracks developed while testing such a cylinder are shown in Fig. 3 and their similarity to the cracks in the tire will at once be noted. The results of this investigation showed that these failures resulted from two causes:

First, lack of uniformity in the ingot from which the tire was made.

Second, the presence of coarse grains or large crystals in the tire.

When a sample containing large crystals suspended in a matrix is subjected to a compression test the large crystals are pressed out of the matrix and when subjected to repeated loads either shell out or develop cracks.

Figs. 4 and 5 show sections of the ingot from which tires are now manufactured and their almost absolute uniformity will be noted from these photographs.

Generally speaking, all results which we have obtained on fatigue tests show that the material of high elastic limit withstands a greater number of rotations under a given fiber stress than material having a lower elastic limit, irrespective of elongation or contraction of area.

The United States Government made a series of extensive experi-

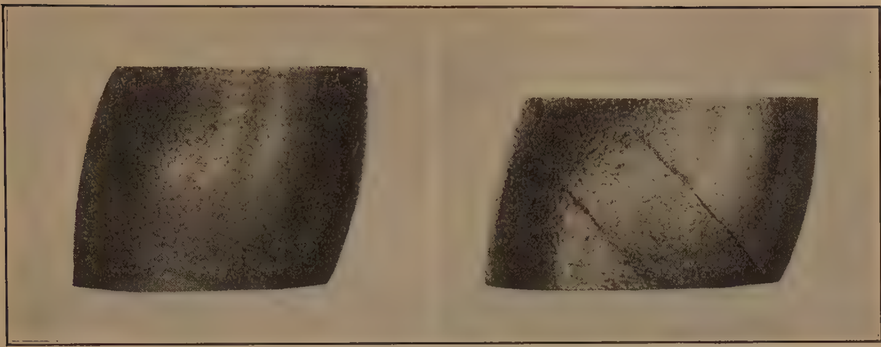

FIG. 3.—TEST CYLINDER SHOWING CRACKS DEVELOPED BY COMPRESSION.

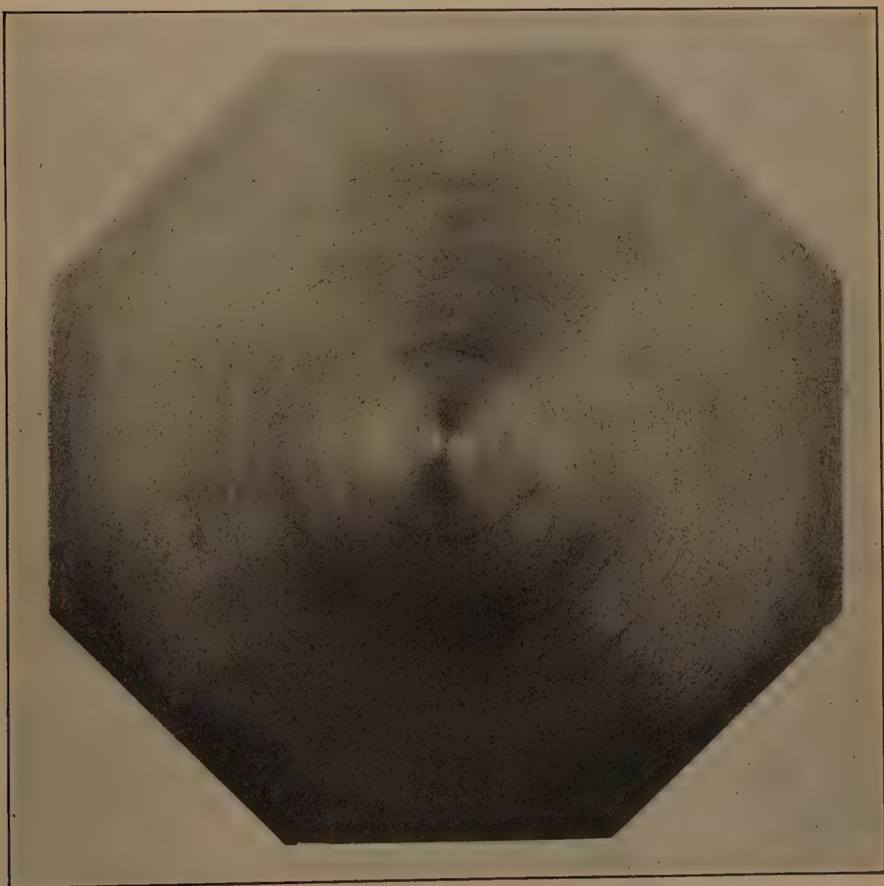

FIG. 4.—SECTION OF INGOT FROM WHICH TIRES ARE NOW MADE.

ments on this subject at the Watertown Arsenal on endurance of rotating shafts, the results of which are given in the following statement:

These results show an enormous increase in endurance of material having high elastic limits and generally confirm the views of Mr. Mitinsky.

I am also giving a statement showing results of endurance tests on steels of different carbon limit loaded with varying fiber stresses and these results likewise show the great endurance of the high-carbon steel or a

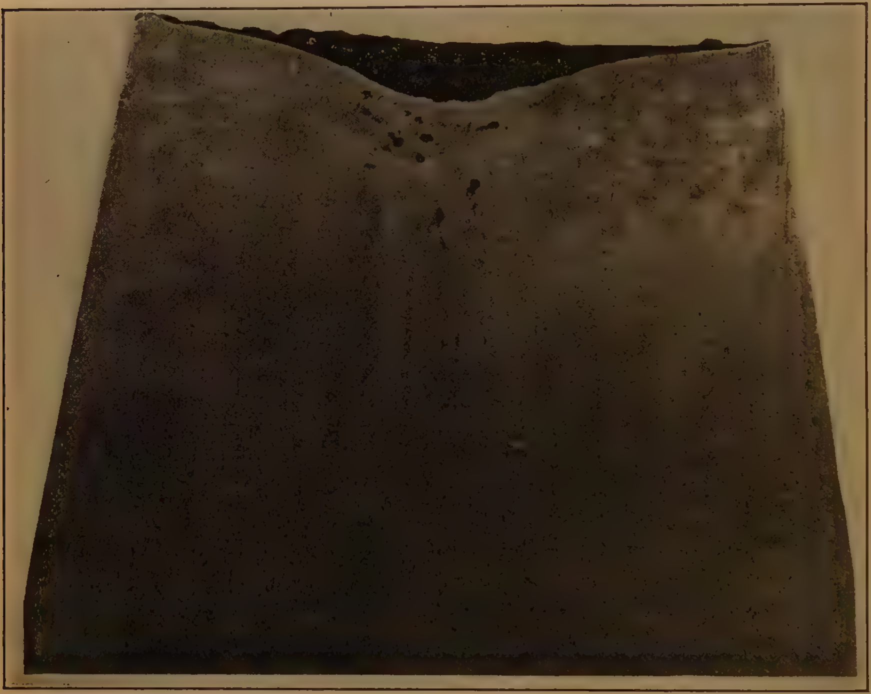

Fig. 5.—Another Ingot Section Showing Uniformity.

steel having a high elastic limit, irrespective of elongation or reduction of area.

In this connection I would say that a method of testing has recently been suggested to determine the endurance value of a sample of steel without consuming a vast amount of time which this test requires.

It has been observed that when a test piece is pulled in a testing machine the temperature thereof falls as a result of the stretching or expansion in the molecules of the sample but when the molecules slide over one another or when a permanent deformation takes place, the temperature of the test piece rises. The load at which this change takes place has been very accurately measured and it is claimed that this value is a measure of the limit of indefinite endurance to repeated stresses. This is no doubt the same value as that referred to by Mr. Mitinsky and it is

very much lower than the point at which a permanent set is first observable.

Carbon	Nickel	Treatment	Fiber Stress	Rotations	Elastic Limit per Sq. In., Lb.	Tensile Strength per Sq. In., Lb.	Elongation Per Cent.	Contraction of Area Per Cent.
0.26	3,282	Annealed	40,000	1,847,500	51,500	81,370	28.75	61.09
0.26	3,282	O., Q & A.	40,000	1,815,200	66,950	90,640	26.90	65.04
0.25	4,514	Annealed	40,000	2,366,000	61,610	95,490	24.80	57.13
0.25	4,514	O., Q. & A.	40,000	3,296,700	101,860	120,190	20.80	60.05
0.29	5,661	Annealed	40,000	4,388,400	80,090	108,840	22.50	58.71
0.29	5,661	O., Q. & A.	40,000	3,795,200	117,610	131,410	19.65	58.88
0.539	27,353	Annealed	40,000	2,495,600	48,060	104,820	47.50	63.10
0.539	27,353	O., Q. & A.	40,000	1,088,200	46,850	97,780	43.35	60.80
0.24	....	Annealed	40,000	229,300	40,560	71,240	32.30	59.81
0.24	....	O., Q. & A.	40,000	348,000	45,170	74,440	33.15	69.93
0.42	....	Annealed	40,000	225,900	44,290	80,885	23.00	56.70
0.42	....	O., Q. & A.	40,000	655,600	55,000	92,180	26.05	57.22
0.46	....	Annealed	40,000	976,600	48,060	94,600	21.15	47.65
0.46	....	O., Q. & A.	40,000	1,657,500	61,110	102,880	23.05	51.27
0.66	....	Annealed	40,000	3,689,000	65,205	124,200	7.15	17.28
0.66	....	O., Q. & A.	40,000	4,323,600	92,040	154,920	13.50	31.48
1.094	....	....	30,000	50,000,000	....	....	....	....
0.733	....	....	30,000	12,547,600	....	....	....	....
0.824	....	....	30,000	16,336,200	....	....	....	....
0.824	....	....	35,000	13,871,000	....	....	....	....
0.094	....	....	35,000	19,152,300	....	....	....	....

Tensile Tests and Repeated Stress Tests on Different Carbon Steels

Description (Steel)	Elastic Limit per Sq. In., Lb.	Tensile St'gth per Sq. In., Lb.	Elongation in 4 In., Per Cent.	Contraction of Area, Per Cent.	Mech'l Work at Rupture per Cu. In, Ft.-lb.	Max. Fiber Stress per Sq. In., Lb.	No. of Rotations at Rupture	Mech'l Work at Rupture per Cu. In., Ft.-lb.
0.17 Carbon	51,000	68,000	33.5	51.9	982	60,000	6,470	32,835
						50,000	17,700	62,635
						45,000	70,400	201,960
						40,000	293,500	665,290
						35,000	5,757,920	9,992,390
						30,000	*23,600,000	*29,500,000
0.55 Carbon	57,000	106,100	16.2	18.7	1,047	60,000	12,490	63,387
						50,000	93,160	328,000
						45,000	166,240	476,900
						40,000	455,350	1,032,130
						35,000	900,720	1,563,125
						30,000	*19,870,000	*24,838,000
0.82 Carbon	63,000	142,250	8.5	6.5	888	60,000	37,250	169,044
						55,000	93,790	399,780
						50,000	213,150	750,465
						45,000	605,460	1,736,910
						40,000	*17,560,000	*40,973,000
						35,000	*19,220,000	*33,635,000

* Not ruptured.

Our experiments likewise show that the size of the crystals rather than the tensile strength is a dominating factor on the anti-abrasive properties of steel. If the crystals are large it is evident that they are easily worn or broken while if they are small it will require a greater time to produce a given amount of wear. Our experiments further show that steel with a large-sized crystal will flow under load more readily than a steel with small-sized crystals and it is evident that the greater number of smaller crystals exhibit a greater resistance than a small number of large crystals.

These results are generally confirmatory of Mr. Mitinsky's experiments.

Henry M. Howe, New York, N. Y. (communication to the Secretary*).—Chief Engineer Mitinsky has put us under obligation by his extremely interesting paper on this very important subject. The total expense to which the community is put by our feeling obliged to make several different reception tests is very large, and could be greatly reduced if we could be confident that a single test were sufficient. The matter is so important that it is to be hoped that Mr. Mitinsky will favor us with a much fuller indication of his authority for his statements concerning the importance of the proportional limit.

Without wishing to deny the eight suggestions with which he closes his paper, I must say that there is much evidence which does not agree closely with what he says. For instance, in the extremely trustworthy work of Ewing and Humfrey[1] on very uniform Swedish iron with a proportional limit of 12 to 13 tons, they find that slip bands appeared after repetitional stress under a fiber stress of 6.9 tons, or not much over one-half of the proportional limit. When it was 7.3 tons, slip bands appeared and broadened in such a way as to suggest that they would in time cause rupture, though the stress was only 58 per cent. of the proportional limit. Finally when the stress was 9 tons or more it invariably led to rupture, although it was not more than 75 per cent. of the proportional limit.

Other evidence showing that the proportional limit is materially beyond the safe working load for repeated stresses is the assertion of no less eminent authority than J. E. Howard.[2] He says that "a load of one-half the tensile elastic limit and less than one-third its tensile strength will in comparatively short time, by means of repeated stresses, rupture one of these rotating shafts."

My own interpretation, which I believe is also that of Mr. Howard, is that the results which he had in mind in writing this were based on material in which the proportional limit had been raised unduly by cool

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* Received Aug. 26, 1915.

[1] The Fracture of Metals under Repeated Alternations of Stress, *Philosophical Transactions of the Royal Society*, A, vol. cc, p. 241 (1903).

[2] *Trans.*, xxiv, p. 777 (1894).

rolling. Mr. Howard's expression "elastic limit" is to be interpreted as referring to the proportional limit.

The proportional limit is very greatly influenced by the finishing temperature and by cold rolling. It is, in my opinion, probable that the elevation caused by cool rolling within ordinary industrial practice is wholly fictitious as regards the endurance. So that in this respect the proportional limit alone is not a trustworthy guide to the endurance.

Mr. Mitinsky's eighth suggestion would, I think, be a closer approximation to the truth if it said that if the total range of stress from the greatest plus to the greatest minus stress does not exceed the proportional limit as determined in a tensile test, no number of repeated stresses will destroy the metal. That is to say, it is rather the total range of stress than the maximum stress reached in either the tensile or compressive phase that counts in tending to break the metal down.

I have yet to see any clear proof that the endurance is more closely related to the proportional limit than it is to either the yield point or the tensile strength. I do not deny that the proportional limit is the most trustworthy of the three, but I do say that I have not yet seen evidence to prove that it is, and that I should value evidence to this effect very greatly.

His statement "the proportional limit cannot however be carried above the yield point"[3] seems to require some explanation. Will he favor us with his authority for this assertion?

While it is true that the artificial raising of the proportional limit in tension by repeated stresses is accompanied by corresponding reduction in the proportional limit in compression, this is not true according to my own observations of single applications of tension and compression. On the contrary, I find that a single application of tensile overstrain is isotropic in its action, strengthening the metal in compression and tension equally.[4]

Mansfield Merriman, New York, N. Y. (communication to the Secretary*).—The conclusions of the author regarding the value of the proportional limit seem valid for static stresses and also for stresses due to loads applied without sudden shocks. For bridge members I agree that the proportional limit is the most important property to be specified or determined by test. Perhaps a sudden shock may be defined as an application of a load which causes local injury to the material. Railroad

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[3] *Trans.*, This volume, p. 163.

[4] Are the Effects of Simple Overstrain Monotropic? *Proceedings of the American Society for Testing Materials*, vol. xiv, 1914. Étude sur les Propriétés Physiques et Mécaniques des Métaux, *Congèrs International des Méthodes d'Essai, Paris Exposition*, vol. i, p. 270 (1900). See also Galy-Aché, *Mémoires, Revue de Métallurgie*, vol. x, p. 587 (1913).

* Received Sept. 7, 1915.

rails are subject to such shocks due to imperfections of track and running gear; moreover the rail itself may have defects due to lack of care in manufacture and these are especially liable to develop further under the action of sudden shocks.

The drop test now used in this country aims to imitate the shocks that occur to the rail in service and also to discover the internal defects of the metal. This test has proved very valuable in insuring better qualities of rails than would otherwise be possible.

I am of opinion, however, that a test for proportional limit ought to be introduced into rail specifications. The arguments of the author in favor of this seem sound except that he does not sufficiently consider sudden shocks. For resistance against such shocks the drop test seems necessary. But under fair conditions of track it is found that the percentage of rail failures is very small, and hence it appears that the quality of the large percentage of rails which do not break might be improved by introducing into the specifications a requirement for proportional limit.

James E. Howard, Washington, D. C. (communication to the Secretary*).—The remarks of Mr. Mitinsky are directed to methods of test having to do with railway material. The special feature, which is peculiar to the Russian specifications for rails, consists in the determination of the elastic limit of the finished shape, inferentially making that value the basis for the working stresses in the track.

The author remarks upon the position held by Hooke's law in the calculations for engineering structures, without which it would be impossible to design such structures intelligently; refers to the elastic limit as the important factor in establishing a limit on the allowable stresses, and the need of fixing the limits of the working stresses with reference to the elastic limit.

Those tests are desirable which furnish exact information on the physical properties of the materials which are used, hence the determination of the elastic limit of the rail commends itself. The acquisition of this information also carries with it the associated idea that equally exact knowledge will be acquired concerning the stresses in the track, in order that the working or track stresses shall bear their proper relation to the elastic limit of the steel. The argument for exact testing loses much of its force if the stresses to which the materials are to be exposed remain undefined.

As a method of test there appears no practical difficulty in determining the elastic limit of the rail. With the introduction of such testing facilities as would naturally follow, the determination of the elastic limit might be done very expeditiously, closely competing with current tests in respect to the total time of handling the material. Tests, how-

* Received Oct. 1, 1915.

ever, are not to be judged according to the time required to make them, but fundamentally from the information which they furnish.

The drop test perhaps has the distinction of furnishing about as limited an amount of information as any which may be mentioned. It has not been claimed for it that it tells the value of the elastic limit. It is not uncommonly supposed to have a relation to service conditions of the rail in the track, although the relation is not as close as at first sight appears.

The striking velocity of the tup of the drop test is much lower than the velocity of high-speed trains. The blow of the tup is one of retarded velocity and, unless the rail fractures, comes to a state of rest at the time of maximum strain. The loading of the rail, as accomplished in the track, is without retardation of speed. So far as known, there is no lag in the display of elastic extensibility in steels by rapidity of loading. From this it would appear that the part of the drop test having to do with the elastic extension of the metal was without special efficacy. The final portion of the test, during which the rail receives a permanent set, constitutes a stage which does not or should not have its counterpart in the track.

It is desirable, however, that information be at hand on the effect of rapidly applied loads, of speeds up to and beyond ordinary train speeds, with loads capable of giving different amounts of permanent sets. Such tests possess a research character, upon which the ordinary drop test has very little bearing. There was occasion in a report of the Interstate Commerce Commission, dealing with a broken rail which caused a derailment in December, 1911, to refer to a test made under static conditions in a testing machine, saying: "This test permitted of the determination of the elastic limit of the rail, an important factor in the strength of materials, but not ascertained in the prescribed drop test." On that occasion it was also shown that the elongation in the drop test was from 5 to 7 per cent., while in the testing machine the rail bent through an angle of 30° without rupture, the extension of the metal reaching 16 per cent. It is the function of a research inquiry to develop the influence of rapidly applied loads taken at different temperatures and the effect of cross-section of rail. It is not established that a limited range in striking velocities, varied according to the weight of the rail, has real significance and furnishes reliable data on the properties of the rail.

These thoughts lead to giving a preference to a test which has the merit of furnishing a well-defined value, and one which in apparently all engineering matters, except rails, is recognized as the factor most needed.

Concerning the sharpness of definition of the elastic limit, that will depend upon the grade of the steel and the rate of cooling and relative thickness of the different parts of the rail. Internal strains in cooling

and differences in the elastic limit of the thick and thin parts of the cross-section will ordinarily result in some vagueness in establishing the elastic limit of the rail as a whole. Failure to recognize influences which tend to cause local sets has led to confusion in judging of the elastic limit of steels. It is well to bear in mind that it has not been shown that any change in the value of the modulus of elasticity occurs until there is an overstraining load on the metal.

Methods of test form only a part, and perhaps a minor part, of the rail question. In fact, it almost seems futile to consider the careful determination of a property of the rail when service conditions so generally result in overloading that particular feature to which attention is directed. Rails present an example in which, very commonly, there is an overload, exceeding the elastic limit, the vital question touching upon the ultimate result and what constitutes the elements of safety in the use of the rails under such circumstances.

The cold rolling by the wheels, when the rail is in the track, introduces internal strains which are evidence that the elastic limit of the steel has been surpassed. This is a matter of common occurrence from which few if any rails are exempt. The higher the wheel loads, the deeper the penetration of their effects, and the more of the metal thus disturbed.

The internal strains introduced attain values not infrequently exceeding the direct bending stresses of certain of the wheel loads. They are strains of compression in the metal immediately next the running surface of the head of the rail, while there are strains of tension in the center of the head which furnish the necessary reaction to the compressive strains. In some recent tests conducted by the Interstate Commerce Commission, in connection with rail failures, compressive strains were measured in rails from service exceeding 21,000 lb. per square inch equivalent stress and over 9,000 lb. per square inch tension in the center of the head. It will be seen that these results explain the formation of the development of transverse fissures in the heads of rails.

These internal strains which are set up by the action of the wheels on the head, on the running surface and on the gage side of the head, do not appear to have attracted attention, although they seem to afford a direct measure of the effects of high wheel loads. In combination with the bending stresses, they reduce the magnitude of the tensile strains in the fibers of the head most remote from the neutral axis of the section, while increasing the tensile strains of the interior fibers. The cold rolling embrittles the steel, as made manifest in the loss of ductility in rails from service when loaded to destruction, with the head in tension.

In a practical way, the greatest interest attaches to such tests as are calculated to insure the acceptance of rails which give the best service in the track. Chemically hard rails resist abrasion in a very satisfactory manner. They have furnished, however, a large percentage of those rails

which have displayed transverse fissures. Tests for acceptance do not cover such cases of failure. Against fatigue failure, by repeated stresses, current tests furnish no data. It would be an aid to have the elastic limit of the rail determined, still the influence of the internal strains from the action of the wheels introduces a factor concerning which there is lack of information.

The development of transverse fissures seems to be the only type of failure due to repeated stresses, the type chiefly or solely due to stresses which approach the elastic limit, in longitudinal tension. To prolong the life of the rail, one or both of the component factors should be reduced, either the section of the rail so increased that the longitudinal tension shall be materially decreased, or the wheel loads reduced to effect a reduction of the internal strains.

In discussing the rail question it is rather difficult to keep close to the matter of specifications and tests. Features not directly pertaining to tests and specifications, as they are being currently carried out, force themselves into prominence and make the tests appear subordinate to other considerations.

On the one hand, high wheel loads and hard steels show a tendency to develop transverse fissures, constituting a track condition; on the other hand, the presence of laminated and seamy steel leading to split heads and base fractures constitutes a mill condition. Those are the vital features which have to do with a large share of rail failures.

It would appear to some that knowledge of the chemical composition of the steel and the finishing temperature of the rail would furnish information from which the elastic limit of the rail might be judged. That information would probably constitute the basis on which the steel maker would proceed in the manufacture of the rails to meet given specifications for the elastic limit. This idea carried out might lead to less testing and more complete knowledge of the processes of manufacture. The fact of making a test implies there is a doubt concerning the product. The test piece does not remove the doubt concerning the next heat of steel, nor furnish absolute data concerning other portions of the heat it is supposed to represent. The principal safeguard lies in the possession of knowledge of the processes of manufacture, in the production of the desired grade and quality of steel; then, since there is an ultimate limit to the endurance of all steels, adopting and providing such conditions of service as the steel is capable of sustaining with safety.

George K. Burgess, Washington, D. C. (communication to the Secretary*).—Mr. Mitinsky's paper, embodying as it does descriptions of test methods so greatly at variance with those practices with which we are familiar in specifications for railway materials, is of very great interest

* Received Oct. 23, 1915.

and value. That it is a summary of some of the methods employed and results obtained by the Department of Tests of the Russian State Railways, gives very great weight to the conclusions.

To define and interpret adequate tests for some of the classes of railway material, such as axles, tires, rails, and wheels, has been and still is a very perplexing matter; and the types of specification for these materials are being constantly questioned and revised.

Mr. Mitinsky concludes "that if a rail or a tire shows a high proportional limit and sustains satisfactorily a drop test, the steel is of as good quality as can be desired;" all other tests, both chemical and physical, he considers superfluous.

This, of course, is in strong contrast with many American specifications for articles of steel, with their rather sharply defined chemical limitations, oftentimes accompanied by manufacturing restrictions and complemented by physical tests of various kinds.

Most of the European countries also have even more elaborate acceptance tests of railway materials such as rails and tires; although, with the exception of the United Kingdom, where for rails, the limits of C, Mn, Si, P, and S are specified, as is the United States practice, the chemical requirements are, in appearance at least, less exacting for rails. Thus, in Italy only the maximum P and S contents are specified, in France C and P or C, or even nothing chemical is specified depending on the railroad, and this last is also the practice of Holland, Austria and Germany, while Sweden specifies only the P content.

There is as great variation among the physical tests of rails required in the several countries, and in the United States, from one railroad to another, in spite of the standardizing societies. There is the practically universal requirement of some form of impact or drop test, which is, in general the only physical test practiced in America. In the European countries, this is supplemented either by a tensile test, bending test, or more commonly by both; and in the case of Italy also by an etching test to eliminate segregation in the rail; while Germany, Austria and Sweden require a hardness (ball) test in addition to the other physical tests.

Mr. Mitinsky does not touch upon the equally important question, which is indissolubly linked to the testing operations for railway material, namely, what discard shall be made of the ingot and how shall the amount of this discard be determined? This has been answered recently by another Russian engineer[1] who describes an ingot practice for which he finds a 6.25 per cent. discard sufficient. Several American specifications require, in effect, the mills to make discards of over 25 per cent. of the usual type of ingot before reaching material that the mill will offer under the specification.

[1] S. Schukowsky in *Organ für die Fortschritte des Eisenbahnw* (1914); also in *Génie Civil* (1914).

Neither Mr. Mitinsky nor Mr. Schukowsky—who is engineer of the Dneiper Works—considers chemical segregation of importance in defining the properties of a rail.

Regarding the rolling of rails, the significant statement is made that "if the rolling of a rail is finished at a high temperature the value of the proportional limit will be low, so that a measurement of the proportional limit will show whether or not the rolling was properly done." If Mr. Mitinsky could offer experimental data on this subject, additional light would be thrown on a phase of the rail problem that has been the cause of considerable differences of opinion in the United States.

If Mr. Mitinsky's premise be admitted, i.e., that the proportional limit is the proper basis of all engineering calculations, it would follow that most of our specifications for metals should be revised.

There is one corollary, however, regarding the properties of material expressed in terms of the proportional limit which does not appear to agree with experience and in support of which direct evidence seems to be lacking in Mr. Mitinsky's paper; namely, "The resistance of fatigue is directly dependent on the proportional limit, and provided this is not exceeded no number of repeated stresses will destroy the metal." If this were true, an alternating stress machine set for less than the proportional limit would run forever on one sample. It might do so if it could operate upon a single crystal of an absolutely pure metal; but with the complex, largely non-metallic structure of the material known as "rail-steel," such generalizations will always break down.

Mr. Mitinsky's figures corroborate well his general thesis, but a more detailed and complete statistical statement of failures or endurance of material in service, tested by the methods he describes, would be of great interest.

The following incident is not without significance: Not long ago, I was showing to a Russian engineer of railways a typical case of the failure known as "transverse fissure" in a rail; his comment was "That could not occur in Russia."

Gaetano Lanza, Philadelphia, Pa. (communication to the Secretary*).—In this paper Mr. Mitinsky makes a strong plea for determining the proportional limit in practical commercial testing, and for basing upon it the value of the working strength to be employed in designing. He also deplores the too general, erroneous custom of confusing it with the yield point, and seeking the latter instead of the former, in order to save time in testing. He points out that the two are very different; that from the one the other cannot be inferred, and that the yield point is not a suitable quantity upon which to base the working strength to be used in practice. He calls attention to the fact that, if

* Received Oct. 25, 1915.

we employ a continuous motion of the machine, a slow speed, and apparatus of sufficient accuracy and delicacy, the experimental determination of the proportional limit need not take more than 15 or 20 min.

As to basing upon the proportional limit the value of working strength to be adopted, I will say that in my opinion its determination ought to rest upon a consideration of the proportional limit and the ultimate strength together, and not upon one of them to the exclusion of the other. Indeed, were the ultimate strength disregarded, it would be possible to so treat the steel as to raise the proportional limit artificially to such an extent that the working strength chosen would be too large a fraction of the ultimate strength to insure safety.

The paper seems to imply that the proportional limit is identical with the breaking strength for repeated and alternate stress. This is probably approximately true for perfectly sound material free from sharp corners and from abrupt changes of section, but in the case of such metal as is in common use, with its ordinary defects, blow-holes, segregation, etc., and with more or less sudden changes of section, the proposition is at least doubtful, and much experimental work on repeated and alternate stresses is needed to determine the actual facts. In the light of our present knowledge it would not be safe to assume that the results of such tests could be predicted without making them.

Albert Sauveur, Cambridge, Mass.—The author expresses the belief that steel will not fail under fatigue when subjected to ultimate stresses unless the stresses exceed the elastic limit or the limit of proportionality. I think this statement is contrary to the evidence. It is conceivable, however, that a stress ever so small does produce a permanent set, although so slight that we cannot detect it with the instruments at our disposal. Nevertheless the repetition of this small stress a great many times will finally break the piece and the nearer we approach what we call the elastic limit, the quicker will the piece break under fatigue.

J. W. Richards, So. Bethlehem, Pa.—I would like to ask Professor Sauveur whether there is a similar indefiniteness about the limit of proportionality as there is about the elastic limit.

Albert Sauveur.—Yes; I believe it also depends upon the delicacy of our instruments.

J. W. Richards.—If that is so then we are as badly off by adopting the limit of proportionality as a criterion as we are by adopting the elastic limit.

William Kent, Montclair, N. J.—I call attention to the proposal made by the late Professor J. B. Johnson in his "Materials of Construction" that the "apparent elastic limit" be defined as that point

on the stress of the diagram at which the elongation per unit of length is 50 per cent. greater than the minimum elongation per unit of length." This gives a point which may be more nearly agreed upon by several different observers than any point corresponding to the other definitions of elastic limit. An equivalent definition is that point at which the quotient of increment of load per unit of section divided by the increment of elongation is two-thirds of the maximum, or in the case of steel of a

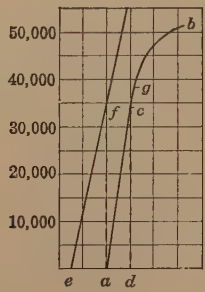

FIG. 1

modulus of elasticity of 30,000,000 that point at which the increase of elongation of an 8-in. specimen per 1,000 lb. per square inch increase of load is 0.0004 in.

To determine this point on the plotted diagram, from some point *c*, well below the apparent limit of proportionality, drop a vertical line *cd*, then *ad* represents the elongation corresponding to the load *c*. Set off $ea = 1.5\ cd$ and $fc = ad$. Join *ef* by a line prolonged above *f*. Then place the edge of a parallel ruler on *ef* and move it until it touches the curve at *g*, which is the "apparent elastic limit."

Conversion Scale for Centigrade and Fahrenheit Temperatures

BY HUGH P. TIEMANN,* B. S., A. M., PITTSBURGH, PA.

(San Francisco Meeting, September, 1915)

THE desirability of employing the centigrade scale for the measurement of temperatures is becoming more and more recognized in this country, particularly in view of the fact that this scale is used almost exclusively in technical papers and publications—exclusively in those published in French or German. There are, however, a large number of factories and laboratories which still employ the Fahrenheit scale. It therefore frequently becomes necessary to convert one into the other and the accompanying chart (Fig. 1) has been found very convenient for this purpose.

The formulas for conversion follow:

$$(1) \qquad °F. = \frac{9°C}{5} + 32$$

$$(2) \qquad °C. = \frac{5\,(°F - 32)}{9}$$

$$(1) \qquad \frac{100°\,C \times 9}{5} = \frac{900}{5} = 180$$

$$\therefore 180 + 32 = 212°\,F.$$

$$(2) \qquad \frac{5\,(212°\,F - 32)}{9} = \frac{5 \times 180}{9} = 100°\,C.$$

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* Metallurgist, Carnegie Steel Co.

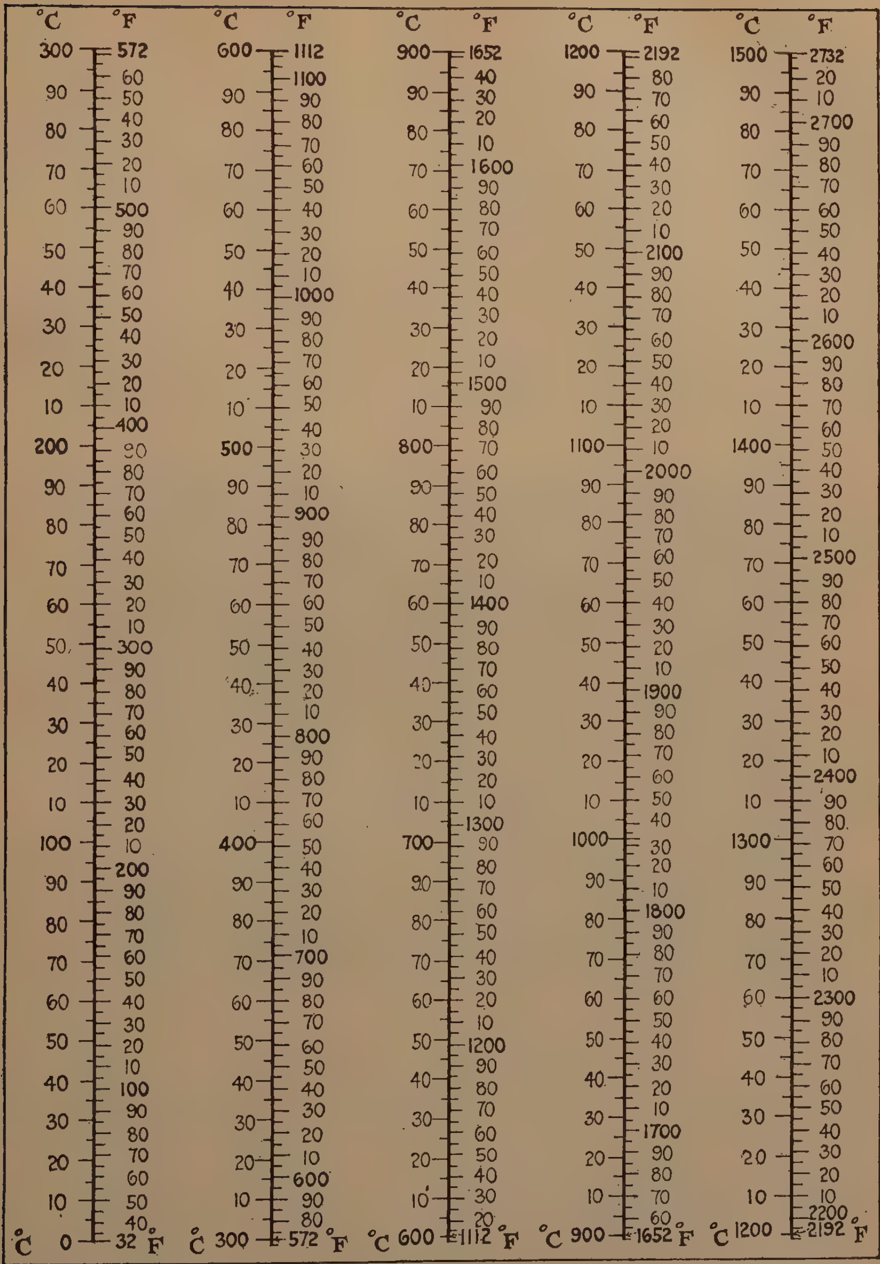

FIG. 1.—CONVERSION SCALE FOR CENTIGRADE AND FAHRENHEIT TEMPERATURES.

Discussion

J. W. Richards, South Bethlehem, Pa.—I have found in practice that among the many ways of figuring from one scale to the other, the following is the easiest to do mentally: Double the Centigrade figure, deduct one-tenth from this (you then have the Fahrenheit degrees above 32°), and finally add 32. Reversely, start by subtracting 32 from the Fahrenheit degrees, then take half, and to this add one-ninth its value.

A curious transformation was discovered by D. H. Childs.[1] To the Centigrade degrees add 40, take five-ninths, subtract 40. To the Fahrenheit degrees add 40, take nine-fifths, subtract 40. The rule is identical for the two cases, except for the inversion of the factor, and it is an interesting bit of arithmetic to show that the transformation is mathematically correct.

Hugh P. Tiemann, Pittsburgh, Pa. (communication to the Secretary*).—This scale was gotten up primarily for personal use and for this purpose was kept on my desk where it would be readily available for one conversion or for a number. For isolated conversions required at considerable intervals of time, however, the ordinary formulæ worked out by pencil and paper should be practically as quick.

If my recollection is correct, there was some correspondence in the *Iron Age* recently in regard to so-called "quick methods" for conversion of temperatures, but it seemed to be the general conclusion that the standard formula: $F = \frac{9}{5}C + 32$, etc., is about as satisfactory as any, and that other methods are nearly, if not quite, as long, and if done mentally would offer the same chance for error.

Concluding his remarks, Professor Richards refers to "a curious transformation." While I do not pretend to be a mathematician, it seems to me that the explanation for the method in question is very simple:

It depends upon the fact that the respective zeros of the two scales are not at the same temperature, and that the same reading is secured only at $-40°$. The method in question, therefore, consists in (*a*) correcting a given reading so that it will be based on this common point (adding 40); (*b*) converting from one kind of degree into the other (multiplying by $\frac{5}{9}$ or $\frac{9}{5}$, respectively); and (*c*) restoring the new reading to the basis of its respective zero (subtracting 40).

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[1] *Metallurgical and Chemical Engineering*, vol. xi, p. 230 (May, 1913).

* Received Oct. 13, 1915.

The Control of Chill in Cast Iron. Considering the Elements Effective in the Manufacture of Malleable Castings and Chilled Car Wheels

BY GRAFTON M. THRASHER,* MICHIGAN CITY, IND.

(New York Meeting, February, 1916)

For the proper discussion of this subject it is necessary to incorporate in this paper the substance of part of an article published by me in *Metallurgical and Chemical Engineering* for January, 1915.

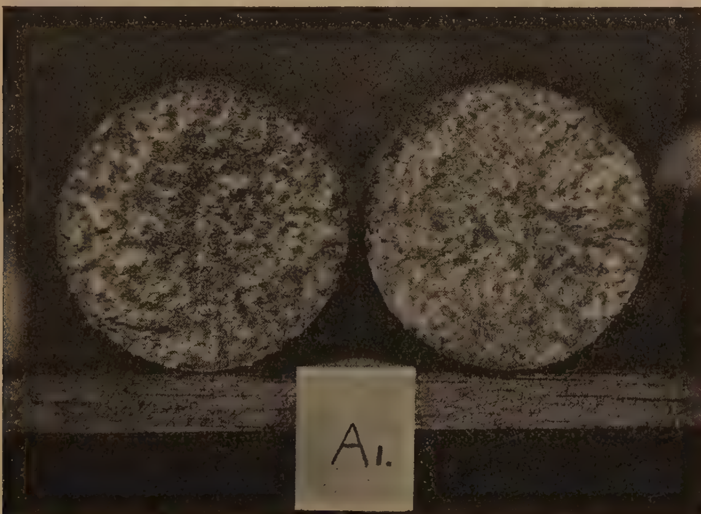

Fig. 1.—Cast-Iron Test Plug, showing Low-Mottled and Mottled Fractures.

Several years ago, while I was chief chemist for a malleable-iron company, it occurred to me that the mere regulation of silicon was not sufficient to produce the desired degree of chill. But only after repeatedly encountering this fact in both open-hearth and air-furnace, as well as cupola metal, was I led to regard variation in the total carbon as the principal cause of disturbance; and I resolved to determine the total carbon as well as the silicon in every heat, and by a carefully prepared test piece to gauge the approximate degree of chill. The test piece selected was a round plug, 2 in. in diameter and 12 in. long, cast on end

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in green sand. It was allowed to cool at least 2 hr. in the mold before being cooled in water and broken in the middle; and the fractures were designated as gray, low mottled, mottled, high mottled, and white. All these fractures except the well-known gray one are illustrated in Figs. 1 and 2. From the same ladle of iron from which the test piece was poured a shot sample was taken for analysis so as to represent the actual iron in the bar as near as possible; and the test in all cases was poured from iron taken from the furnace when about half emptied.

It may be well to mention here that the principal disturbing influence on these tests is the fact (proved by experiment) that it is entirely possible to pour a mottled test and a gray test from the same ladle of iron, depending upon the pouring temperature.

In order to study the cause of the variations more systematically a

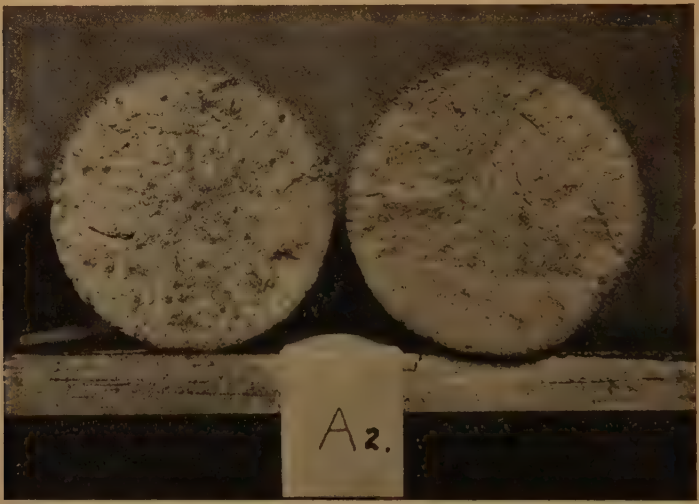

Fig. 2.—Cast-Iron Test Plug, showing High-Mottled and White Fractures.

chart was prepared for plotting the analyses with carbon as ordinates and silicon as abscissæ. It soon became apparent that the analyses corresponding to the test piece with average chill, designated as "mottled," were arranged along a diagonal line on the chart, with analyses corresponding to high and low chill at a more or less definite distance on either side of the line. It happened that about that time a heat had to be held pretty long in the furnace on account of a breakdown, and it became necessary to doctor it up with 50 per cent. ferro-silicon to get the metal into pouring condition. The test piece from this heat was found to be just normal or what is designated above as "mottled," and the corresponding analysis was silicon 1.07 and carbon 2.47 per cent., which served admirably for one extreme of the chill line as drawn on the chart. Something to represent the other extreme was found in a

pig of charcoal iron of the same dimensions as the test piece and having the proper degree of chill.

More recently I have obtained data for a new high-silicon point in which the analysis was, silicon 1.25, carbon 2.25 per cent.

The chart as finally constructed is shown in Fig. 3.

As the manganese in all of these heats ranged between 0.25 and 0.35 per cent., which is normal for air-furnace malleable, and as sulphur averaged about 0.06 per cent., it will be seen that these elements interfered but little with the results. The phosphorus, of course, was between 0.13 and 0.18 per cent. and likewise had very little influence.

In examining the chart it will be noted that while the line tends

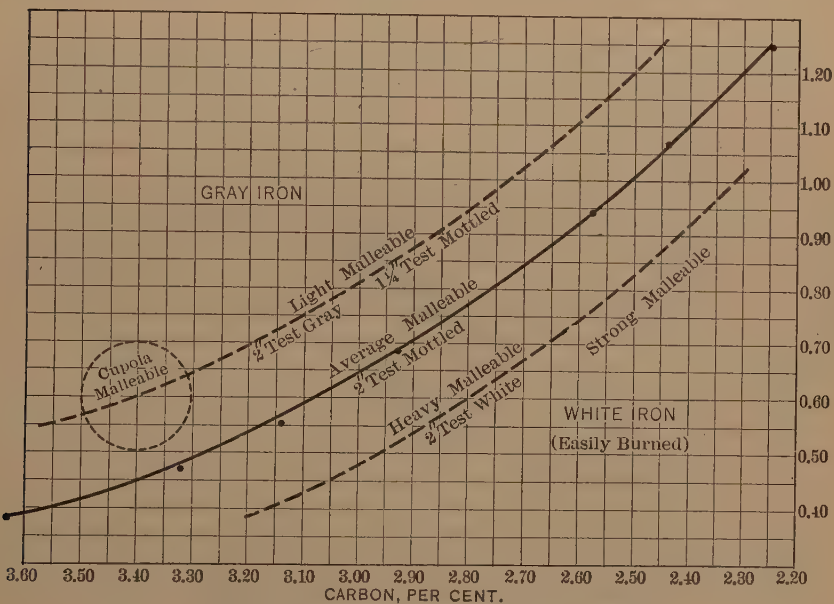

FIG. 3.—LINES OF EQUAL NATURAL CHILL.

to curve somewhat, yet it is sufficiently straight through the usual range of analyses to permit the assumption for practical purposes that 4 points of carbon are equivalent to 3 points of silicon in the regulation of chill, when the silicon is above 0.70 per cent.; and that below this about 2 points carbon are equivalent to 1 point silicon.

The value of this knowledge will be at once apparent to a malleable-iron foundryman, as it enables him to substitute large quantities of low-carbon material for his sprue in case of shortage of that material, thus:

It is desired to substitute steel containing silicon 0.10 and carbon 0.30 per cent. for 1,000 lb. of sprue containing silicon 0.80 and carbon 2.80 per cent.

Equivalent deficiency of silicon in the steel due to low carbon = (2.80 − 0.30) × ¾ × 1,000	= 18.7 lb.
Silicon deficiency = (0.80 − 0.10 × 1,000 =	7.0
Total equivalent silicon deficiency........	25.7

Therefore, it is necessary to increase the silicon in the mixture 25.7 lb. by substituting higher-silicon pig iron.

The popular conception of malleable iron accepts the limits of 0.60 to 0.90 per cent. in silicon, with a very indefinite notion as to what the carbon content in the hard state should be; but a careful examination of this chart will show that for a given silicon percentage there is a definite carbon percentage, which is proper for each class of castings.

In order to prevent cracked work and for other economic reasons, the malleable-iron foundryman runs his mixture as nearly gray as he dares, but if he desires to produce a very strong iron the presence of primary graphite in the hard iron must be avoided by lowering either the silicon or the carbon or both.

As to malleable iron produced with the same average chill but with varying analyses, it will be found that the high-silicon, low-carbon mixture is more sound and reliable, but possibly a little less fluid, while the high-carbon, low-silicon mixture is subject to greater shrinkage and segregation, which produce weakness.

The principal consideration for the foundryman is to produce that type of structure which is necessary to obtain malleability upon annealing: and the chart shows the proper adjustment of silicon and carbon to bring about this desired result for the different grades of work.

It will be further noted that, of the three factors considered, namely, silicon and carbon content, and chill, if any two be known the other may be predicted approximately.

Malleable iron produced from the cupola is pretty well confined to the dotted circle shown on the chart. Its proximity to gray iron, as well as its position in the field of greater shrinkage, will account for its weakness, which is not due as popularly supposed to the higher sulphur content. As the carbon is automatically adjusted within this range by melting in the cupola, the regulation of chill must be effected by silicon alone. It is not feasible, however, to run a mixture lower than 0.50 per cent. in silicon because the cutting action of the slag thus produced is very hard on cupola linings. Good malleable castings are produced along the whole length of the charted lines; but the average foundryman, with his limited knowledge of conditions to be met in another foundry, cannot see how the other party makes a certain casting with 0.90 silicon while he finds it necessary to make it with 0.60 silicon. It is hoped that the publication of this chart will clarify the matter. The lines here

given have been confirmed by plotting hundreds of analyses of mixtures of known chill and normal pouring temperature and the chart has been found a valuable aid in practical work.

Regarding the strength of malleable iron produced from irons of varying degree of chill, the following data, charted in Fig. 4, from test bars poured every other day for two weeks, will give a good idea of the effect of this factor:

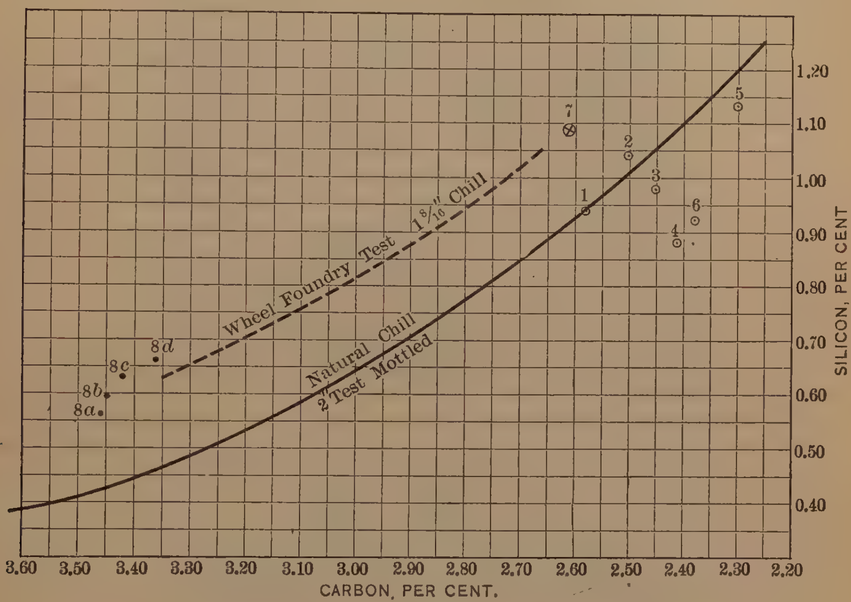

FIG. 4.—ANALYSES OF BARS, TESTED FOR TENSILE STRENGTH.

Bar No.	Tensile Strength, Pounds per Square Inch	Elongation, Per Cent.	Silicon, Per Cent.	Carbon, Per Cent.
1	43,680	10.0	0.94	2.58
2	44,770	9.5	1.04	2.50
3	48,340	10.5	0.98	2.45
4	49,710	13.0	0.88	2.41
5	49,440	12.0	1.03	2.30
6	48,080	11.0	0.92	2.38

These test bars were round, 0.8 in. in diameter for 2 in. in the middle, and enlarged at the ends for gripping. A shrink ball was cast on one end to insure soundness. The structure of two of these bars is shown in Figs. 5 and 6. No. 1, the weaker bar, shows slight inclination to graphitic structure in the shape of the carbon particles.

The nearest approach that I have noticed to the co-ordination of

such valuable facts as are above outlined is the paper published in the *Proceedings of the American Foundrymen's Association*, a couple of years

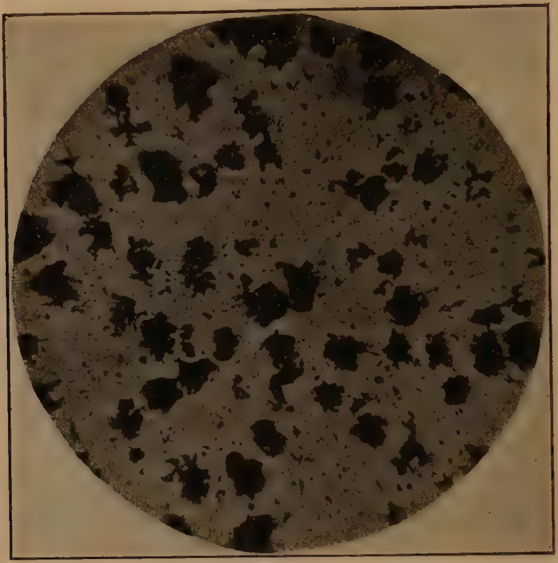

FIG. 5.—STRUCTURE OF BAR NO. 1, MAGNIFIED 50 DIAMETERS.

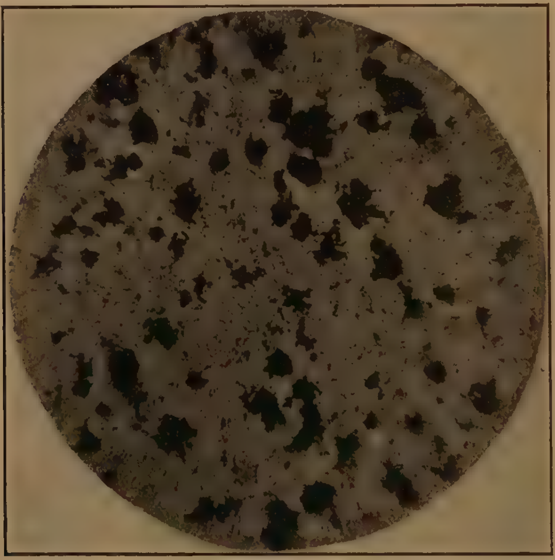

FIG. 6.—STRUCTURE OF BAR NO. 4, MAGNIFIED 50 DIAMETERS.

ago, by John Jermain Porter, in which he plotted silicon and combined carbon; but as he made no statement as to the size of the castings from which these results were obtained, it is probable that this factor

was not kept constant, which of course would account for a great many irregularities. Also, no reference was made to the effect of the total carbon on the combined-carbon content, which, as may be seen above, is highly important.

For the purpose of comparing the "natural chill" with that produced when cast iron is poured against an iron chiller, a test was made which conformed to that used in a car-wheel foundry, using the same pattern and chill box. A ladle of very hot low-carbon iron was treated with sufficient 50 per cent. ferro-silicon, as calculated from the chart, to produce the same degree of chill as that of the iron used in car wheels. It was necessary to allow the metal to cool for several minutes in the ladle before pouring, in order that the temperature should approximate the practice in the wheel foundry, where the metal is not so hot when melted. The result of this experiment is charted in Fig. 4 as No. 7, while a regular test piece from a wheel mixture showing the same degree of chill ($1\frac{8}{16}$ in.) is charted as No. 8. The significance of this experiment is that it points the way to the production of car wheels with high silicon, low carbon, and low sulphur by the use of an open-hearth furnace for melting. The rapidly increasing sulphur in the cast-iron car wheel of today, due to the remelting of old wheels, is a source of anxiety to both the manufacturer and the railroad. The percentage of cementite in the structure of such a metal would still be sufficient to insure good wearing qualities and a higher percentage of silicon would insure soundness. The absence of large quantities of manganese sulphide, which is undoubtedly the cause of a large percentage of the defects in the tread, would be a strong argument in favor of such a metal.

It may be profitable to consider here the relationship of the "lines of natural chill" as described above to the line of saturation of silicon and carbon in cast irons. For this purpose I have prepared another chart, Fig. 7, which outlines this relationship. The saturation figures are taken from Keep's *Cast Iron*, p. 45, and from the work of Wüst and Petersen, as outlined by Hatfield.[1] The line I produced, which is that of "natural chill," is the same as that marked "2″ Test Mottled" in Fig. 3. As a speculation it has been extended toward 6.67 carbon, as its probable trend in that direction, and upward to 3.50 silicon and 1.25 carbon, where it most probably tapers off and disappears due to the absence of carbon in sufficient excess of the pearlitic ratio to form graphite, which is necessary to produce the mottled fracture described. I hoped to produce by direct experiment a more definite end point on the high-silicon side before presenting this paper; but lack of time prevented.

It may be well to add a word concerning the influence of the other elements necessarily present in commercial irons which have been disregarded in this paper.

[1] *Cast Iron in the Light of Recent Research*, p. 31 (1912).

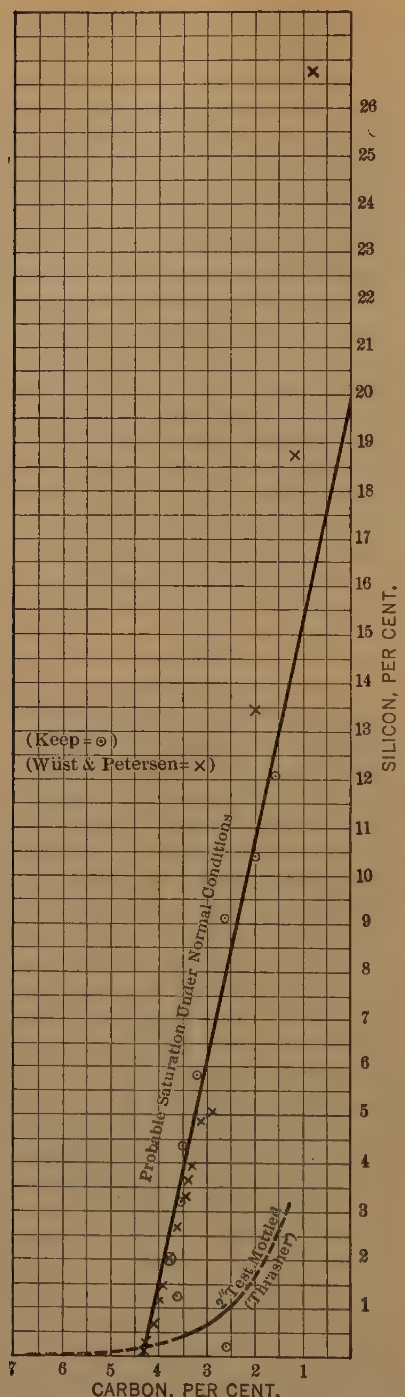

FIG. 7.—RELATION OF LINE OF NATURAL CHILL TO THAT OF SATURATION.

The effects of manganese and sulphur must be considered together, since it has been shown by several eminent investigators that where present together in the proper ratio they exist as MnS, which, properly distributed, would be neutral and ineffective upon the chill or strength of the metal. In practical work it is found necessary to carry an excess of manganese above this ratio, to insure reliable metal. In cupola-melted irons where the carbon is high, an excess of from 0.05 to 0.10 per cent. of manganese is sufficient, while iron melted in the air furnace or open hearth requires from 0.10 to 0.30 per cent. excess on account of greater exposure to oxidizing influences. It will be seen that where iron is properly melted to a fluid condition, the sulphur is fixed and without

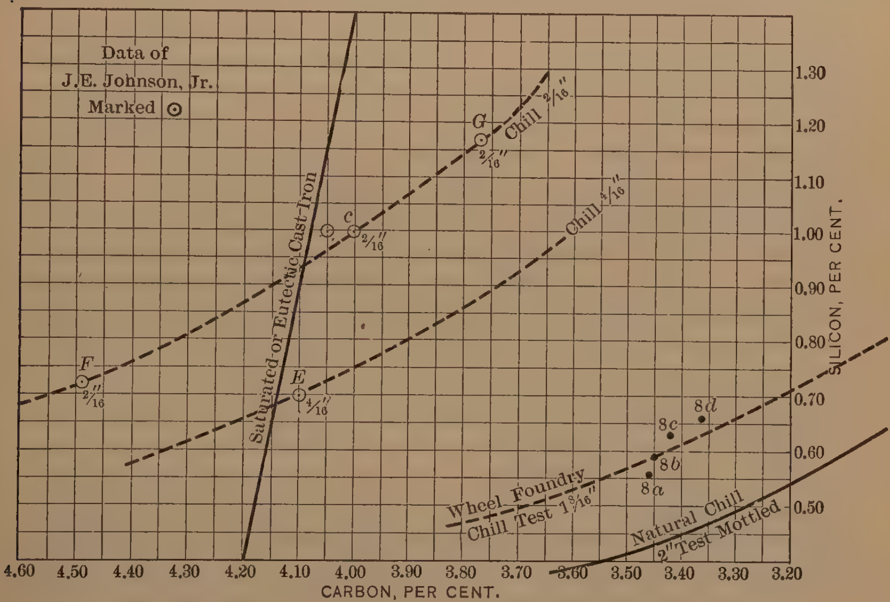

FIG. 8.—DATA OF J. E. JOHNSON, JR.

influence on chill, etc. If the 0.30 per cent. excess manganese is figured to Mn_3C it will be apparent that it can affect only 0.02 per cent. of carbon, which would have but small influence on the chill.

In my opinion the effect of sulphur on the chill, even when present uncombined with manganese, has been greatly exaggerated, and oxygen occurring simultaneously will be found responsible for this exaggeration.

Regarding the effect of phosphorus on the chill, it will be sufficient to cite Hatfield, who says, in discussing Stead's work on this element:

"Since, however, there is seldom more than 1.5 per cent. of phosphorus present in pig irons used in foundry work, it may be considered

that the influence of such percentages upon the condition of the carbon is negligible."[2]

It will be seen that in the control of chill in malleable iron and car-wheel mixtures, where this element does not exceed one-fifth of 1.5 per cent., its effect cannot be noticed.

Though I have not made an independent investigation of the effect of oxygen on the chill, I am inclined to believe from the data at hand that its effect is mainly qualitative rather than quantitative. While I fail to agree with some of the conclusions of J. E. Johnson, Jr.,[3] in his paper on this subject, I think his theory of the existence of an oxy-sulphide is true. In fact, a close analysis of Mr. Johnson's data brings out more forcibly my contention, namely, that the quantitative control of chill in commercial cast irons, aside from the physical effects of casting temperature, size of test, and size of chiller, is practically dependent upon the percentages of silicon and total carbon present. In support of this statement, a chart, Fig. 8, is offered, in which Mr. Johnson's data are tabulated. His *A* and *B* series, not having a measurable chill, have been left out. It will be noticed that his *G*, *C*, and *F* series, forming a line for ⅛-in. chill, and his *E* on a line representing ¼-in. chill, are in perfect agreement with my data, shown in the lower right-hand corner, and all lines verge toward the point representing 6.67 carbon and no silicon. While his specimen *D* does not seem to agree with this theory, it is entirely possible that it may not be in its proper position, either because of error in analysis, making silicon or carbon too low, or because the sample was not representative; though a larger test pig than usual, or very low pouring temperature, would cause the specimen to show no chill.

In concluding I wish to express my acknowledgments to members of the Institute, and especially to Professors Howe, Sauveur, and Stoughton, whose work on cast irons has been helpful and inspiring in the study of this interesting subject.

Discussion

Richard Moldenke, Watchung, N. J.—Mr. Thrasher is to be commended for developing a very interesting comparison between high- and low-carbon and silicon malleable cast irons for an equal degree of chill. His deductions are unquestionably correct, though the application to daily practice is open to serious criticism, from the standpoint of the man who receives the castings, as will be shown later on.

Running over some of the points which affect the tests in question: Mr. Thrasher admits that the temperature of pouring has a serious influence on the degree of chill shown in the test plugs he made. This is

[2] Hatfield: *Cast Iron in the Light of Recent Research* p. 54 (1912).
[3] *Trans.*, l, p. 344 (1914).

a well-known fact, and I would like to add that another serious trouble comes from oxidized iron when the heats have been unduly prolonged. Such metal always shows more chilling action for a given silicon and carbon content than normally melted iron. Indeed, the test piece quoted giving the extreme of the chill line (1.07 per cent. Si and 2.47 per cent. total carbon) was such a poor heat, a ferro-silicon addition never "doctoring" up the metal sufficiently to show normal good strength.

In spite of these irregularities, which would simply change the curve slightly, I take it that Mr. Thrasher's results can be accepted as reliable enough to establish the relationship of silicon and total carbon as shown. When, however, the claim is made that silicon and carbon can be substituted for each other in mixture-making (a claim which may be perfectly correct metallurgically), a very important factor is forgotten, causing the purchaser of the castings much annoyance. This is the contraction (erroneously called shrinkage) of the castings. They will no longer be true to pattern. The lower the total carbon is allowed to drop, the greater the contraction, and increasing the silicon correspondingly will not correct this trouble. No wonder that recently, in revising the Standard Specifications for Malleable Castings it was necessary to increase the variation of castings from requirements considerably over what was formerly found sufficient. The habit of running up the proportion of silicon and escaping danger from grayness of the metal by copious steel additions was evidently too tempting to manufacturers, as their molding losses were thus kept down and a very soft malleable iron resulted. The customer, however, got serious variations in his work, which was not the case when the total carbon was kept between 3.25 and 2.75, and the silicon held as low as possible without undue cracking of the hard castings.

I have been repeatedly called in professionally where it was necessary to have very strong iron with the slightest allowable variation in dimensions. Failure in this was regularly traced to large steel additions to the mixture when the silicon in the heats was increasing.

In my book on the production of malleable castings I have stated that the lower limit for total carbon in important work should be 2.75 per cent., and the silicon held low enough to be certain that even the heaviest castings that may be furthest away from the furnace should show a white fracture. If this is followed, very high-grade work will be produced under proper melting practice. Hence, while Mr. Thrasher's interesting tests are certainly highly valuable, the practical foundryman had better confine his compositions to very small variations of both silicon and total carbon, in order to avoid trouble with his customers.

I further fully agree with Mr. Thrasher in his view that "sulphur effects on the chill have been greatly exaggerated, and that oxygen occurring simultaneously will be found responsible for this exaggeration."

Finally, where Mr. Thrasher quoted the theory of J. E. Johnson, Jr., as to the existence of an oxy-sulphide, I wish to recall the fact that this theory was advanced by Dr. J. J. Porter before the American Foundrymen's Association in June, 1910, as an elaboration of my own claim that the oxidation of iron during the cupola melting process was responsible for many of the troubles iron castings were heir to, and for which no good reason had been advanced theretofore.

Albert Sauveur, Cambridge, Mass.—This is a very instructive paper. I think, however, that the author overlooked some recent views concerning the influence of sulphur on chill, by which it is held that some of the sulphur is present combined with carbon associated with iron carbide, and that its presence there increases the stability of the carbide and therefore opposes the formation of graphite. In this way the sulphur would promote the chill.

Magnetic Studies of Mechanical Deformation in Certain Ferromagnetic Metals and Alloys

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(New York Meeting, February, 1916)

THE application of other than mechanical methods to the study of the mechanical-physical properties of metals has become in the last few years a topic of investigation of ever-increasing interest, both from the theoretical standpoint and from that of the engineer and constructor. Besides the investigations which have been made of the magnetic properties of ferromagnetic metals, in their relation to the mechanical properties of these metals, which will be discussed more in detail below, there have been made studies of other phenomena in the same relation, and methods for the mechanical testing of metals have even been suggested; for example, the microscopic method of Tammann,[1] and the thermo-electric method of Turner[2] for determining elastic limits.

The application of all such methods, which are based upon the dependence of some physical, crystallographic, or possibly chemical property, upon the mechanical state of the metal, must be preceded by close and careful correlation of these various "test" properties with the known mechanical state of the different metals or materials. It is with this view that the present work, as well as that of many previous investigators of, what may be termed in the interests of brevity, the mechanical-magnetic properties of ferromagnetic materials of construction, has been done.

Previous Investigations of Mechanical Magnetic Properties

The effect of mechanical stresses and of deformation, elastic or otherwise, on the magnetic qualities of iron, steel, and other metals, has been studied by a number of investigators from the time of Joule (1841) to the present. It will, however, for the present purpose be sufficient to give an account only of the principal results obtained in the course of these investigations.

It was early noticed (Villari, 1865) that if an iron wire is placed in a

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[1] *Zeitschrift für Elektrochemie*, vol. xviii, p. 584 (1912).

[2] *Proceedings of the American Society of Civil Engineers*, vol. xlviii, p. 140 (1902).

magnetic field of low value, the values of the magnetic induction, measured while the wire is subjected to tension, will first increase with the tension, reach a maximum value and afterward decrease. This phenomenon is called the "Villari reversal," after the discoverer, and was later studied and verified by Sir William Thompson (1877 to 1879).

In a general investigation, covering in its scope the study of the magnetic characteristics of iron and steel and the effect of temperature and of mechanical stress upon these characteristics, Ewing made the first systematic contribution to our knowledge of the mechanical-magnetic properties of iron. Ewing worked with iron and steel wires; he studied the effect of tensile stresses upon the form of the *B-H* curve (the curve giving the relation between the magnetizing force, *H*, and the total induction, *B*); he noticed that iron which had already been stretched beyond the elastic limit (yield point) was more sensitive magnetically to stress than before, and showed that a permanent deformation or set resulted in a permanent decrease in the magnetic permeability. He also studied rather minutely certain magnetic-mechanical hysteresis effects, which will be referred to later in the present paper.

In 1902, Fraichet in a short investigation presented a new aspect of the matter, namely, the applicability of a magnetic method to the study of the mechanical properties of iron and steel. Fraichet placed his specimens in a testing machine, applied a constant magnetizing force, and proceeded to carry out the ordinary tensile test on the specimen, through which the change of magnetic flux or induction was continually indicated, but not measured, by a galvanometer in circuit with a secondary coil surrounding the specimen. He obtained in this way two curves, one, the ordinary stress-strain diagram, and the other, a curve indicating the changes taking place in the permeability of the material of the specimen as a function of the tensile stress to which it was subjected. He observed that the yield point and elastic limit were indicated on the latter curve by wide fluctuations of the galvanometer reading.

Smith and Sherman (1914) have made measurements, using the Burrows compensation method, of the permeability of certain steel specimens during the tensile test, with a view to determining the effect of tensile stress on the form of the *B-H* curve. Their results confirm those of the older investigations, as well as our own, in showing the Villari effect.

Goerens (1912 and 1913), in a comprehensive study of the effect of cold working on the various physical properties of iron and steel, has included studies on the magnetic properties. He found that the effect of cold working was to diminish the permeability, which was restored, however, upon annealing.

Burrows (1912 to 1914), in a number of articles, discusses the correla-

tion of mechanical and magnetic properties, and suggests using the measurements of the latter to determine the former.

The Authors' Investigations

The work, of which a partial account will be given here, was done in 1912 and 1913 at the Königliche Technische Hochschule in Charlottenburg-Berlin.[3] It was considered that the important work of the previous investigators had not solved, or in some cases even considered, some questions which are of vital interest from a practical standpoint; for example, the question as to the magnetic definition of elastic limit, or in other words, the determination of the magnetic change which takes place at that value of the stress called the true elastic limit.

Another question, apparently left unsettled by previous investigation, was the effect of variations of the mechanical stress below the true elastic limit upon the magnetic properties for zero load, or in other words, the question as to whether the magnetic properties of iron and steel are dependent upon their previous elastic history. The value of such a relation, if it existed, is obvious, for it would then be possible, in the investigation of "failed" or other ferrous material, to determine by the aid of magnetic measurements what were the values of the stresses to which the specimen had been previously subjected.

The investigations of Ewing mentioned above have conclusively shown that, provided an iron or steel specimen is not brought to what may be called its normal state by a process of demagnetization before each magnetic measurement, the values of any magnetic quantity so measured are dependent upon the previous mechanical history, even if the stresses applied were below the elastic limit. The utilization in any practical way of the value of a magnetic quantity measured without previous demagnetization is, however, extremely limited, for the reason that there is lag, or hysteresis, in the relation of other variables than stress to permeability; this is true, for example, with magnetizing force. It has been found, however, that the effects of this hysteresis or lag can, in the latter case, be removed by demagnetization, to such an extent that the permeability is a determinable function of the field strength. The question then that remains is whether a stress application within the elastic limit affects the magnetic quantities measured after removal of the stress and demagnetization.

It was intended, further, to give particular attention to the discovery of a possible relation of the position of the permeability maxima in the permeability-stress curves to the values of the mechanical constants, such as yield point and elastic limit.

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[3] A full account of the investigation was published in January, 1914, as a dissertation of the University of Berlin.

So far, the investigation has been limited to the study of the magnetic properties as affected by mechanical tension, and has consisted in simultaneous measurements of magnetic and mechanical quantities of the materials during the tensile test. The magnetic measurements were carried out on the specimen in the testing machine.

Description of Materials and Methods Used

To avoid complications as much as possible, materials were chosen for the investigation which were homogeneous in structure and thoroughly annealed at the outset. The annealing insured that the material was free from initial stress of any sort which would tend to obscure the results obtained. That this annealing is necessary is shown in a comparison of results obtained on material as received and as afterward annealed.

The principal material used was a soft iron of low carbon; a wrought iron, a sample of nickel, and a nickel steel were also used. The analyses of these metals are shown in Table I.

Table I.—*Metals Used in Tests*

Number	Material	Analysis, Per Cent.
1	Soft iron	C, 0.08; Si, 0.04; P, 0.008; Mn, 0.23; S, 0.026.
2	Wrought iron	....
3	Nickel	Fe, 1.70; Co, 0.77.
4	Nickel steel	C, 0.45; Ni, 20.40.

These materials were furnished in the form of rods, which were turned down to the test shape desired, 50 to 55 cm. in length and from 0.8 to 1.20 cm. in diameter, and then annealed in an atmosphere of nitrogen, or *in vacuo*, at a temperature of 950°C. This was done in a long electric furnace specially constructed for this work. The specimens were allowed to cool in the furnace, unless otherwise noted.

In outline, the measurements were conducted as follows: The test bar, inserted through a specially constructed magnetizing coil, was placed in the testing machine. An extensometer was attached, which measured the elongation immediately adjacent to the part of the test bar included within the magnetizing coil. Stresses of various values were imposed, and magnetic and extensometer measurements taken.

The Magnetic Measurements.—Only a brief description of the apparatus used in the measurements of the magnetic quantities is given here. For further details reference should be made to the dissertation previously mentioned. The total inductions, B, were measured ballistically in the usual manner; the values of the field strength, by the aid of the mag-

netic-potential-meter (*Spannungsmesser*) described by Rogowski and Steinhaus.[4]

In order to determine extremely small changes in the magnetic induction, the latter was not measured directly, but in each case the difference between the permeability of the specimen under stress and that of a similar specimen of the same material, outside of the testing machine and under no stress, was measured. In this manner it was possible to measure changes in the induction of from 0.2 to 1.0 gauss,

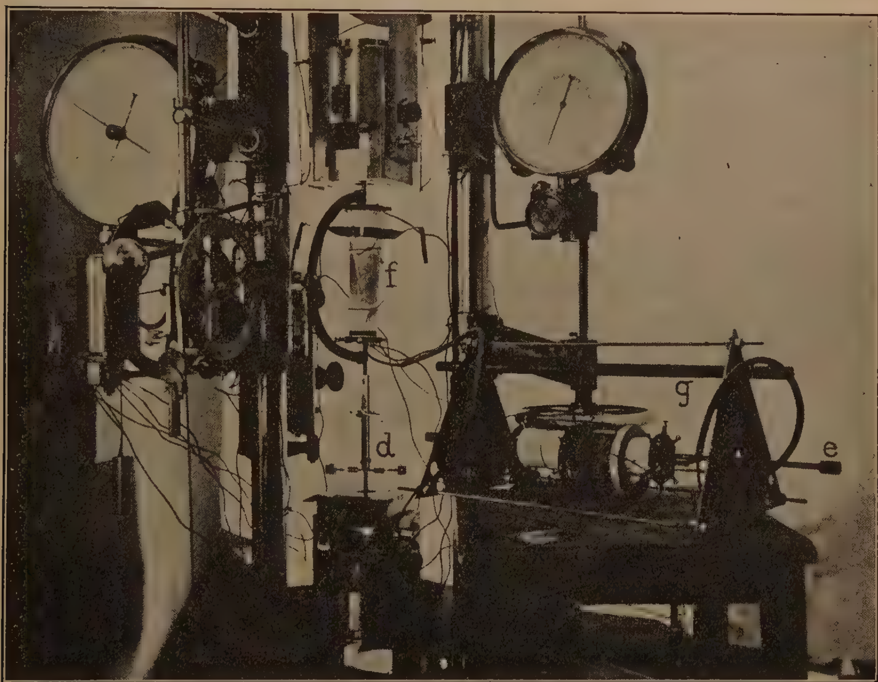

FIG. 1.—TENSILE-TESTING MACHINE EQUIPPED TO MAKE SIMULTANEOUS MAGNETIC AND MECHANICAL MEASUREMENTS

depending upon the value of the total induction. The measurement of the absolute value of the permeability itself was probably subject to a maximum error of about 2 to 3 per cent.; that of the variations of the permeability with applied tension, to a less error.

Physical Measurements.—The physical tests were made in a 5,000-kg. Losenhausen universal testing machine. The elastic elongations were measured with the well-known Marten's mirror extensometer. The elastic limit will be defined here as that least value of the stress at which there remained a permanent elongation of 0.0002 per cent., since variations, in the test length of 15 cm., of this amount could be measured,

[4] Archiv für Elektrotecknik, vol. vi, p. 141 (1912).

and successive determinations of the length at zero load agreed to within this amount.

Fig. 1 shows the whole apparatus as set up in the testing machine ready for a series of measurements. The test specimen is shown in the jaws of the testing machine, with the spool *f*, containing the magnetizing and secondary coils, suspended independently, and with the mirror extensometer *d* attached. The compensating bar *e* is shown in the yoke *g*.

At the beginning of a series of measurements, the specimen in the machine was demagnetized and the *B-H* curve directly determined for zero load. Thereupon the magnetizing and secondary coils of the comparison bar *e* were thrown into circuit in such a way that the ballistic throw of the latter was in the opposite direction to that of the secondary of the test bar in the machine, almost wholly neutralizing it when the test bar was not stressed. Both bars were now demagnetized by the same current, the magnetizing force removed, and a stress applied to the test bar. Both bars were again demagnetized and the difference between the values of the total induction in the stressed and in the unstressed comparison bar directly determined, for various values of the field strength, in the usual way. This process was always adhered to; any change in the stress applied to the test bar was made only after demagnetization and removal of the magnetizing force, and was always followed by demagnetization of both bars. The temperature of the bars was kept constant during the measurements by a cooling coil, with running water, which was made a part of the magnetizing spool *f*.

Results and Discussion

Types of the curve series obtained are shown in Figs. 2 and 3. These show the total induction *B* for constant field strengths as a function of the applied tensile stress (abscissas) for the soft iron (material 1) as annealed, and as annealed and elongated 5.7 per cent. in the testing machine, respectively. The curves have maxima, which occur at large values of the stress at low values of the magnetizing force *H*, and lesser values of the stress with increasing values of *H*. For higher values of *H*, these maxima disappear entirely.

The positions of these maxima, that is, the values of the stress at which they occur in each case, are different in the annealed and in the stretched or elongated material. The maxima occur in the elongated material at higher values of the stress than in the case of the annealed material before it has undergone any plastic deformation. It is well known that certain mechanical constants occur also at higher values of the stress in cold worked or stretched than in soft or annealed material, i.e., the elastic limit, the yield point, and the ultimate strength. The

question presents itself as to what relation there is, if any, between the displacements of these magnetic maxima in the induction and the displacements of these latter mechanical constants, as iron is cold stretched or elongated.

In order to study this question, several series of measurements were made, of which the results are shown in Fig. 4. A test specimen was

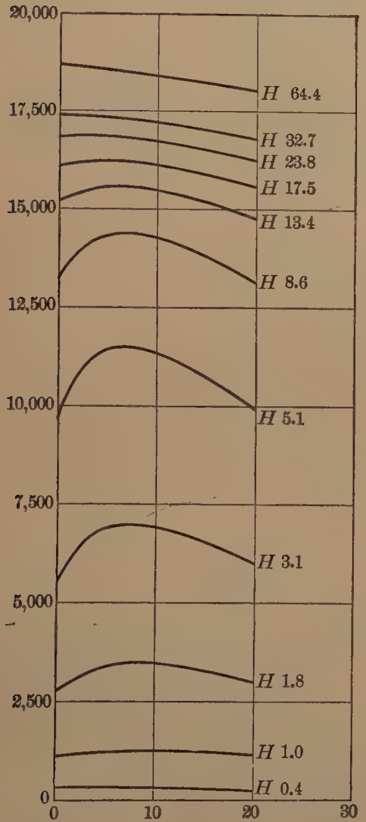

FIG. 2.—MATERIAL 1—IRON, ANNEALED. ABSCISSAS—STRESSES IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—INDUCTIONS IN GAUSSES.

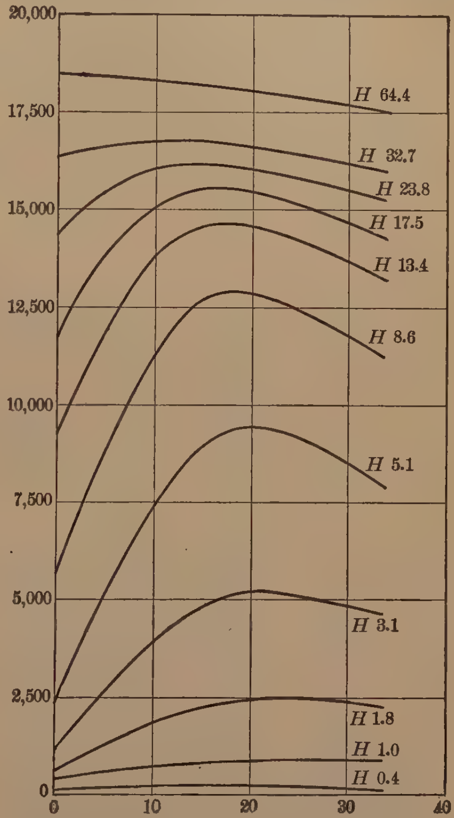

FIG. 3.—MATERIAL 1—IRON, ANNEALED AND 5.7 PER CENT. ELONGATED.

loaded up to and beyond the yield point, unloaded, loaded again up to and beyond the new yield point, unloaded, reloaded, etc. Each time the specimen was loaded beyond the yield point it was elongated permanently by about 2 per cent. The values of the induction B for four values of the field strength and the physical constants (elastic limit and yield point) were determined throughout the series of measurements. The total elongation of 5.8 per cent. was made in three steps, viz., 2.1,

3.7, and 5.8 per cent. After each of these deformations, the induction curve of the material for each value of H followed a different curve, giving with the original, or virgin curve, four in all; these are shown in Fig. 4.

The elastic limit of this material was 14.0 kg. per square millimeter, and the yield point, 20 kg. per square millimeter in the annealed condition. The induction curve for the field strength 3.4, for example,

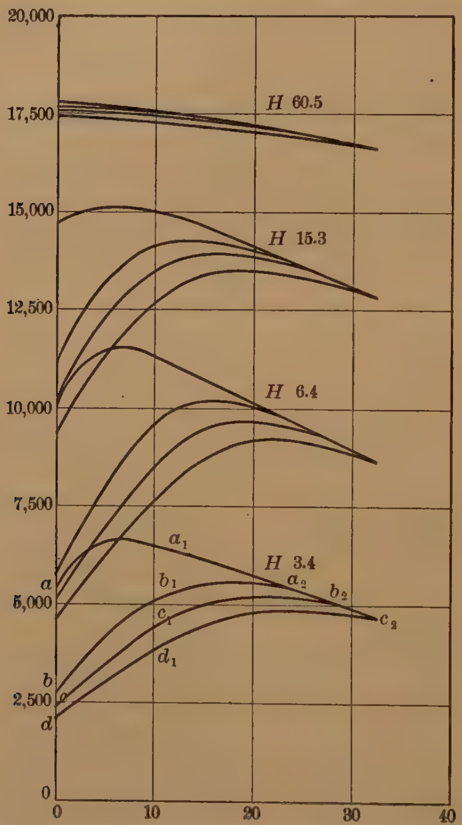

FIG. 4.—MATERIAL 1—IRON, PREVIOUSLY ANNEALED. ABSCISSAS—STRESSES IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—INDUCTIONS IN GAUSSES.

followed reversibly the curve aa_1 up to the elastic limit, and beyond the yield point to a_2, at which point the specimen had been elongated by 2.1 per cent. Upon unloading, the curve a_2b_1b was followed as well as upon reloading. The stress was then increased to the value b_2, whereby the specimen was further elongated by 1.6 per cent. Upon unloading, the new curve b_2c_1c was followed. The load was again applied (curve $cc_1b_2c_2$) and removed after a further elongation of 2.1 per cent.; the induction followed the curve c_2d_1d. The material was therefore tested

in four "states," i.e., after different degrees of homogeneous cold working, after which both the mechanical and the magnetic constants were different.

It can be seen from the curves that as the material is stretched, and the elastic limit and the yield point are raised, the positions of the induction maxima are also raised, for all values of the field strength. In Fig. 5 is shown the relation between the elastic limit and the positions of the induction maxima, for the four values of the field strength. That these are closely related is more strikingly shown in Fig. 6, in which are plotted as abscissas the values of the elongations undergone (a measure of the cold work put on the specimen), and as ordinates, the quotient of the position of the induction maxima (value of the stress at which the

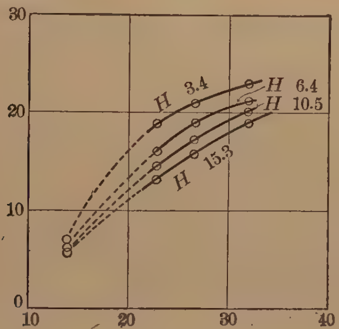

FIG. 5.

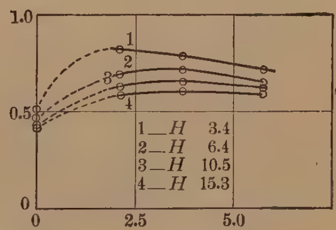

FIG. 6.

FIG. 5.—MATERIAL 1—IRON, ANNEALED. ABSCISSAS—ELASTIC LIMITS IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—POSITION OF THE MAXIMA IN THE INDUCTION IN KILOGRAMS PER SQUARE MILLIMETER.

FIG. 6.—MATERIAL 1—IRON, ANNEALED. ABSCISSAS—ELONGATION PER CENT. ORDINATES—QUOTIENT OF THE POSITION OF THE INDUCTION MAXIMA DIVIDED BY THE ELASTIC LIMIT.

maxima occurs) divided by the value of the elastic limit. Although this quotient is not a constant, it is nearly so after the material has received an initial elongation. The quotient for these field strengths is about 0.45 for the annealed material, and is raised to 0.6, and above, by the first elongation of 2.1 per cent., remaining thereafter approximately constant.

It is interesting also to notice that, whereas there is a large change in the permeability under zero stress after elongation, there is practically none during elongation under the stresses producing it, the curve of inductions and stresses showing no discontinuity, or even marked change of direction, at the yield point.

Careful measurements were made to determine the exact minimum value of the stress at which a permanent change in the induction under zero load occurred, after removal of the load, and to relate this to the elastic limit (the minimum value of the stress at which a permanent

change in the length occurs). A specimen was subjected to cyclical loading, between the limits A and zero, until the test length included by the extensometer remained constant to within less than 0.0002 per cent. A, the upper stress limit, was increased by steps of 2 kg. per square millimeter until a permanent set was obtained. The values of the induction for two values of the field strength were also measured differentially each time after removal of the load. The results of these measurements are shown in Fig. 7, in which are plotted as abscissas the stress, and as ordinates the values of the induction B for the field strengths 34.5 and 5.2. The elastic limit lay, as nearly as could be determined, at 14 kg. per square millimeter, and the yield point at 23 kg. per square millimeter. It can be seen from the curves, that the permanent change

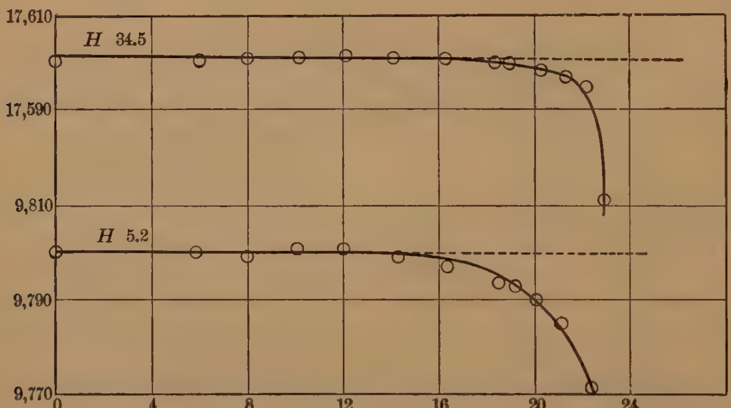

FIG. 7.—MATERIAL 1—IRON, ANNEALED. ABSCISSAS—STRESSES IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—INDUCTIONS IN GAUSSES.

in the induction begins in both cases (for both values of the field strength) at values of the stress lower than the yield point, and that, in the case of the induction for the lower magnetizing force, this change begins at about 14 kg. per square millimeter, the elastic limit. It is probable that differential measurements of the magnetic induction in this way would afford a more sensitive criterion of a plastic deformation in iron or steel than measurements, in the usual way, of the length or elongation.

It is well known that after iron has been stressed beyond the yield point, and thereby elongated, it does not regain, for some time, its elasticity, the elastic limit becoming practically zero. The iron in this temporary state will show hysteresis in its stress-strain curve; the loading curve and the unloading curve do not coincide. Measurements were made to determine whether the magnetic induction was, under these circumstances, subject to the same hysteresis. For this purpose a specimen was loaded and unloaded cyclically, and at each step the

inductions B were measured for several values of the field strength. It was found that as long as the material under test possesses an elastic limit, and the upper value of the stress applied was not greater than this limit, no difference or hysteresis in any part of the induction cycle for a constant field strength could be detected; any such difference was not in any case greater than from 5 to 7 gausses, a matter of 0.07 per cent., which was of the same order of magnitude as the error of measurement in

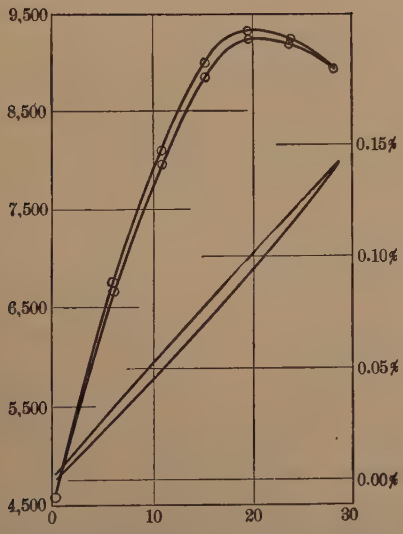

FIG. 8.—MATERIAL 1—IRON. ABSCISSAS—STRESSES IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—INDUCTIONS IN GAUSSES; ELONGATIONS IN PER CENT.

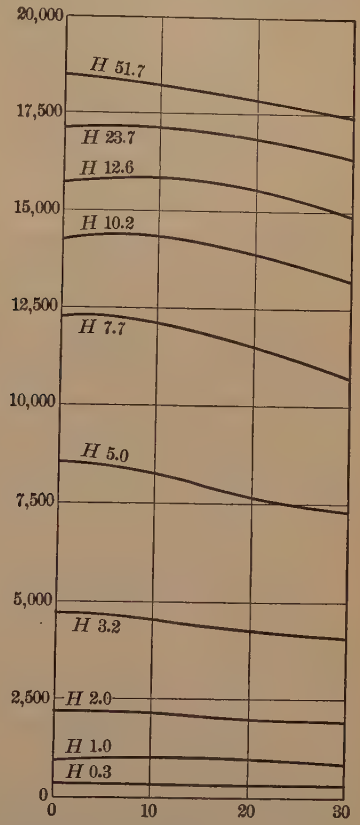

FIG. 9.—MATERIAL 1—IRON AS DELIVERED, HOT-ROLLED. ABSCISSAS—STRESSES IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—INDUCTIONS IN GAUSSES.

this case. The induction cycle, as well as the cycle of elongations, was perfectly reversible. When, however, the iron was stressed over the yield point, and then immediately tested, both the elongation and the induction curves showed hysteresis. The results of one such test are shown in Fig. 8, in which the abscissas represent the stresses and the ordinates, in one case, the elongations, and in the other, the magnetic

inductions B for loading (the "up" curve) and unloading (the "down" curve). These curves show that the maximum lag in the elongation is about 0.005 per cent., and in the induction, about 1.0 per cent. This bar was tested again, in the same way, 24 hr. after being overstrained; the hysteresis of both kinds had completely disappeared.

The results presented and discussed in the last two paragraphs show that, at least for the material tested, the magnetic induction (and permeability) is not dependent upon the previous elastic history of the

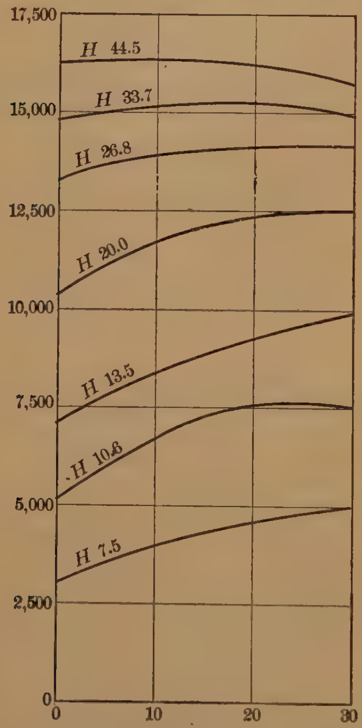

FIG. 10.

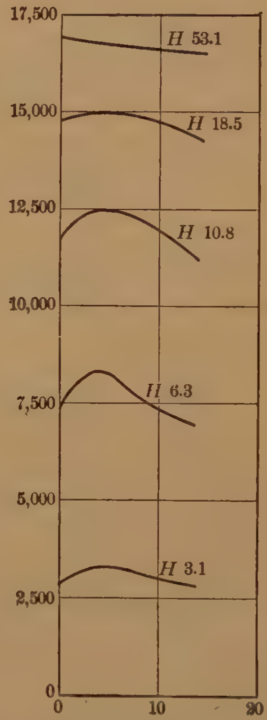

FIG. 11.

FIG. 10.—MATERIAL 1—IRON, QUENCHED FROM 950°C. ABSCISSAS—STRESSES IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—INDUCTIONS IN GAUSSES.

FIG. 11.—MATERIAL 2—WROUGHT IRON. ABSCISSAS—STRESSES IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—INDUCTIONS IN GAUSSES.

material, as long as (1) the variations or alternations of stress have been within the elastic limit, and (2) the number of such alternations has been low. The present study did not include the study of the effect of thousands or millions of stress cycles upon the magnetic constants, as was originally intended. The magnetic method is, therefore, apparently not suited, as it was hoped that it would be, for the determination of the mechanical history of a steel or iron specimen, so long as the stresses in such specimens have been at all times within the elastic limit. If the elastic limit has

been exceeded, and in particular if the yield point has been exceeded, the possibility is given of subsequently determining this fact by the aid of magnetic measurements, such as have been here described.

It was suspected that, since iron after having been stressed beyond the yield point is in a metastable state, indication of this fact would be given by measurements of the magnetic permeability. Specimens were therefore so stressed, the stress removed, and the permeability, or induction, measured for certain values of the field strength, after various intervals of time. One bar showed, for certain medium values of the field strength, a decrease of induction of from 40 to 70 gausses after 21

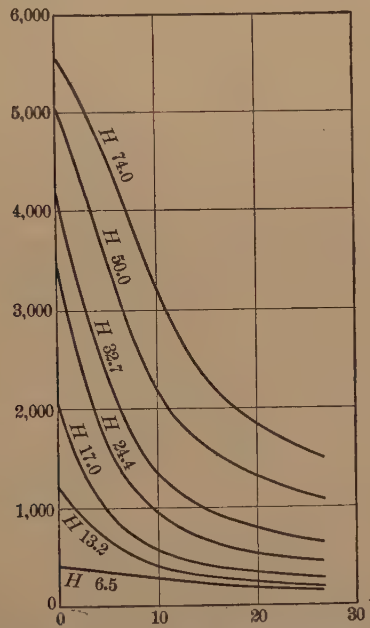

FIG. 12.—MATERIAL 3—NICKEL. ABSCISSAS—STRESSES IN KILOGRAMS PER SQUARE MILLIMETER. ORDINATES—INDUCTIONS IN GAUSSES.

hr.; and the same total change after 24 days. The magnetic change in an overstrained iron specimen takes place, therefore, in that interval of time in which the iron becomes again elastic, i.e., in the period of elastic recovery.

In Figs. 9 and 10 are shown curves illustrating the effect of stress upon the inductions B in the material 1, as delivered (hot rolled), and as annealed and quenched in water, respectively. It can be seen that, in the case of the material as delivered, there is great irregularity in the induction curves. Often the maxima occur at higher values of the stress for high than for low values of H. In both cases, the material is in a state of internal stress due to the manufacture and heat treatment, and it is

to the presence of these stresses that the irregularities in form of the curves are to be ascribed. Strangely enough, the maxima in the induction curves in the latter case are displaced toward higher stresses than in the case of the same material annealed, although the elastic limit is much lower. The elastic limit of the material 1, as delivered, was about 28 kg. per square millimeter. Owing to the presence of these internal initial stresses, it is impossible to trace any such relations between the magnetic

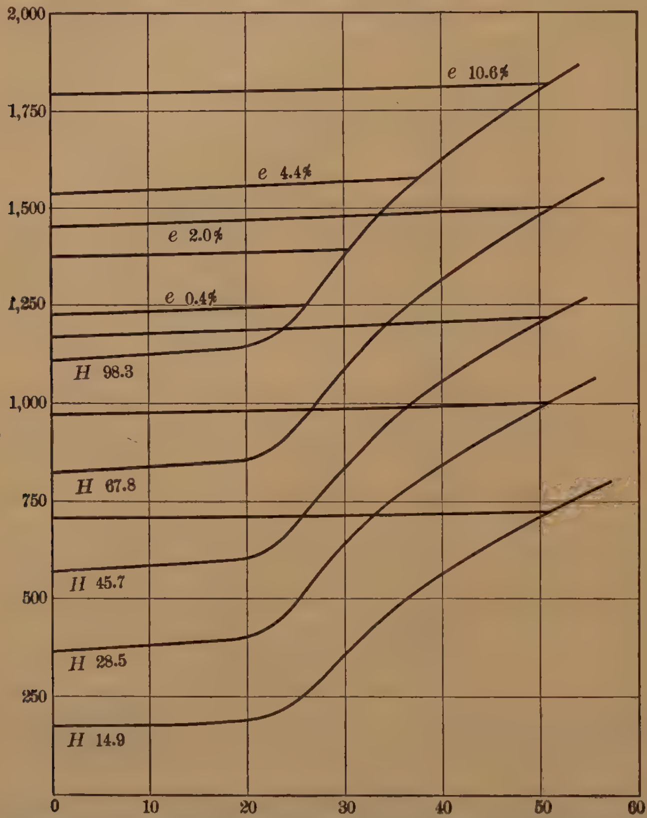

Fig. 13.—Material 4—Nickel Steel. Abscissas—Stresses in Kilograms per Square Millimeter. Ordinates—Inductions in Gausses.

and the mechanical quantities as were noticed in the case of the annealed material.

In Figs. 11, 12, and 13, are shown similar curves for materials, 2, 3, and 4, respectively, as annealed. The induction maxima, in the case of the wrought iron, occur at lower stress values than in the case of material 1, and the elastic limit lies also lower, at about 11 kg. per square millimeter. This again bears out the contention that there is a more or less simple relation between these quantities.

The two latter cases, in which no maxima occur, are interesting. The elastic limit and yield point fell together, at 23 kg. per square millimeter, for the nickel steel; the elastic limit for the nickel specimen, at 24.8, kg. per square millimeter, and the yield point at 30 kg. per square millimeter.

The position of the yield point and elastic limit is shown nicely in the induction curves of the nickel steel by the bend in the curve. The induction curve, for stresses within the elastic limit for this material, annealed, and either before or after elongation, is linear, with a slight slope upward. When the elastic limit is reached, there is a marked increase in the induction, which remains after removal of the stress.

Résumé

1. Simultaneous magnetic and mechanical measurements were made in the tensile-testing machine on a soft iron and other materials. These were made with a view to determining whether relations existed between the physical and the magnetic constants of such materials.

2. It was found that the permeability of iron is independent of the previous elastic history of the material if (1) the alternations or cycles of stress have been within the elastic limit, and (2) the number of alternations has been small.

3. It was found that there is a close and nearly linear relation between the elastic limit of soft iron and the values of the stress at which the permeability maxima occur, as the material is elongated.

4. The magnetic indications and accompanying features of mechanical overstrain were studied.

5. The conclusion is reached that the magnetic method is applicable to the study of the previous mechanical history of such annealed iron as was investigated, since indications are thus given of the values of the maximum stress to which it has been subjected, provided that this stress has been above the elastic limit. The permeability of this material is not affected by stress applications under the elastic limit.

6. The effect of initial stresses upon the form of the induction-stress curves was noticed in the case of material as received, before annealing, and as heat treated by quenching in water.

Discussion

John A. Mathews, Syracuse, N. Y.—I should like to ask the authors if there is any evidence of chemical change resulting in formation of alpha iron from the gamma crystals found in nickel steel of high nickel content when stressed beyond the elastic limit, similar to the effect produced by drastic cooling. Would the stresses mentioned and the drastic cooling both result in an increase in magnetic induction?

I would also like to ask the authors if they have made a test of the hardness. It would be very interesting to see whether the hardness decreased under the conditions of stress, as might be the case if alpha iron were produced.

Paul D. Merica, Washington, D. C.—While it is impossible to state absolutely, it is my belief that the effect of deformation is approximately the same as that of drastic cooling, but we did not make a test of the hardness.

Leonard Waldo, New York, N. Y.—This paper is a very important contribution to the remarkable series of papers originally published by Brown and Hadfield in the *Transactions* of the Royal Dublin Society on the effect of chemical constitution and heat treatment on steels designed for loading coils and ballast coils, on which our whole modern long-distance telephone depends. The results here given show what was there anticipated, namely, that one of the most important effects on high inductions was produced by mechanical treatment. I wish the analyses were more complete. I should like to ask whether No. 3 in Table I is pure nickel or nickel steel.

W. E. Ruder, Schenectady, N. Y.—That is nickel.

John A. Mathews.—I did not quite grasp Dr. Merica's remarks in reference to the difference between fine-grain and coarse-grain steels as applied to the slip-band theory. Can it be proved that the total area of the slip bands in fine-grained steels is greater than the area of slip bands in coarse-grained steels, even though the area of the individual slip bands in isolated crystals would be smaller?

Paul D. Merica.—In the case of steel which has been plastically deformed somewhat severely, slip bands are formed, representing planes of discontinuity in the material, which, together with the original grain walls, form a greater total area of planes of discontinuity than in the original annealed material. Now we know that the presence of such surfaces increases the resistance of a material to magnetic or other flux, and indeed according to the Tammann-Heyn theory of plastic deformation the decrease of permeability in a steel after plastic deformation is to be explained by the formation of such planes. If pressure is applied perpendicularly to such a plane tending to press the two opposite surfaces together, the contact resistance of the surface is diminished, and, conversely, if tension is applied to this surface tending to pull the opposite surfaces apart the resistance must be increased.

My point is, therefore, that if plastic deformation is to be explained entirely by the formation of gliding planes, as claimed by Tammann and Heyn, the increase of permeability upon applying tension to an annealed steel with few surfaces of discontinuity certainly should not

be less than that caused by the application of the same tensional stress to the same steel, plastically deformed. The fact that this increase is less, and markedly so, is submitted as an indication that the above-mentioned theory is not sufficient to explain the change of properties caused by the plastic deformation.

Leonard Waldo.—What is an elastic stress?

Paul D. Merica.—It is a stress that does not produce a permanent set, so far as we can observe it.

John A. Mathews.—Do you think there is a feasible method of determining the elastic stress by magnetic arrangement?

Paul D. Merica.—It is certainly feasible, but I think the Martin extensometer is much more convenient.

Leonard Waldo.—It would be a great comfort to know that Martin's extensometer is in use in the United States. I was very much impressed with the very great certitude of the results under stress afforded by the reflecting extensometer of Martin. So far I have never met it in practical works and I am very glad indeed that the Bureau of Standards has set the example for its use.

The Effect of Carbon on the Physical Properties of Heat-Treated Carbon Steel*

BY J. H. NEAD,† B. S., RACINE, WIS.

(New York Meeting, February, 1916)

Object

The experiments herein described were undertaken with a view to investigating thoroughly the influence of carbon on the tensile and impact physical properties of carbon steel. The original comprehensive plan included investigation of a series of steels with varying carbon contents over a wide range of heat treatments. These were to be studied as to their static tensile qualities, and also their resistance to shock as measured by the Charpy impact bending test. A thorough metallographic examination was likewise included in the schedule. The Charpy tests and the metallographic examination have not been completed to date, and are accordingly withheld for another report. All heat treatments, however, have been made, and the static tensile tests have been completed. A description of the procedure of the experiment and the results thus far obtained are therefore reported herein.

Conclusions

No attempt is made to draw general conclusions from these experiments, but the curves and the data are submitted as being of general importance and wide applicability.

Composition of Materials

An attempt was made to secure steels increasing in carbon by increments of 0.1 per cent. from 0.1 per cent. up to 0.8 per cent., and from that composition to 1.6 per cent. by increments of 0.2 per cent. Actually, however, steels of the composition given in Table I were procured, and as the carbon contents were arranged sufficiently well so that satisfactory curves could be drawn showing the relation between carbon content and other properties, these steels were used in the experiments.

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* This work was done while the author was Metallurgical Engineer at Watertown Arsenal, Watertown, Mass.

† Metallurgical Chemist, Belle City Malleable Iron Co.

TABLE I.—*Chemical Composition of Steels Used*

Carbon, Per Cent.	Manganese, Per Cent.	Sulphur, Per Cent.	Phosphorus, Per Cent.	Silicon, Per Cent.	Chromium, Per Cent.
0.14	0.45	0.035	0.018	0.131	None
0.18	0.56	0.043	0.024	0.132	None
0.32	0.51	0.027	0.009	0.128	None
0.46	0.40	0.050	0.020	0.144	None
0.49	0.60	0.028	0.013	0.127	None
0.57	0.65	0.028	0.012	0.167	None
0.71	0.67	0.035	0.027	0.147	None
0.83	0.55	0.028	0.018	0.152	None
1.01	0.39	0.029	0.016	0.160	None
1.22	0.34	0.031	0.025	0.181	None
1.39	0.20	0.029	0.015	0.191	None
1.46	0.20	0.035	0.011	0.133	0.35

It will be noted that the carbon content varies from 0.14 to 1.46 per cent. The manganese content varies from 0.20 to 0.67 per cent., the amounts of this element being normal for commercial steels. It

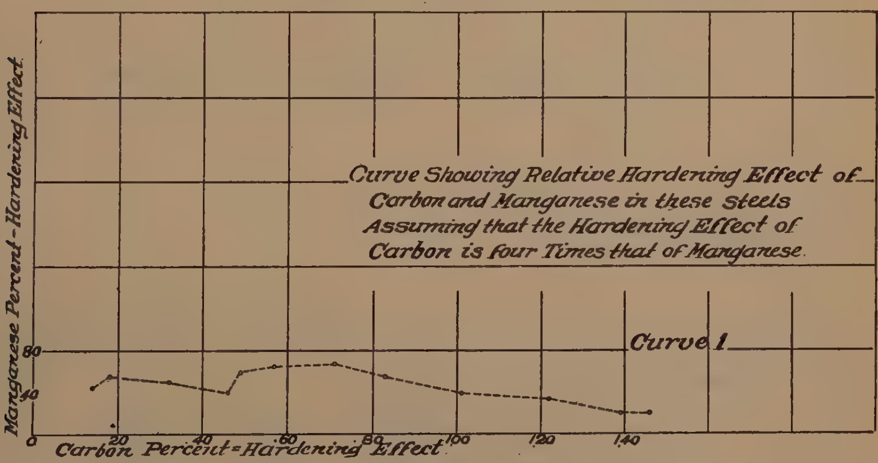


FIG. 1.

was thought that the variations in manganese content were not great enough to disturb seriously the conclusions to be drawn as to the effect of carbon.

Fig. 1 is an empirical curve showing the relative values of the hardening properties of the carbon and the manganese in these steels. The ordinate scale is reduced to one-quarter that of the abscissa scale, arbitrarily assuming that the hardening value of manganese is one-fourth that of carbon. This is an assumption often made in practice. The coördinates of any point on the curve represent, then, the relative harden-

ing effect of the carbon and the manganese in the steels whose compositions are given in Table I.

Heat Treatment

No critical point determinations were made on these steels. The temperatures used in the heat treatments were based on the recommended annealing temperatures for rolled and forged carbon steel objects of the American Society for Testing Materials.[1]

The recommended temperatures were plotted in the form of a curve,

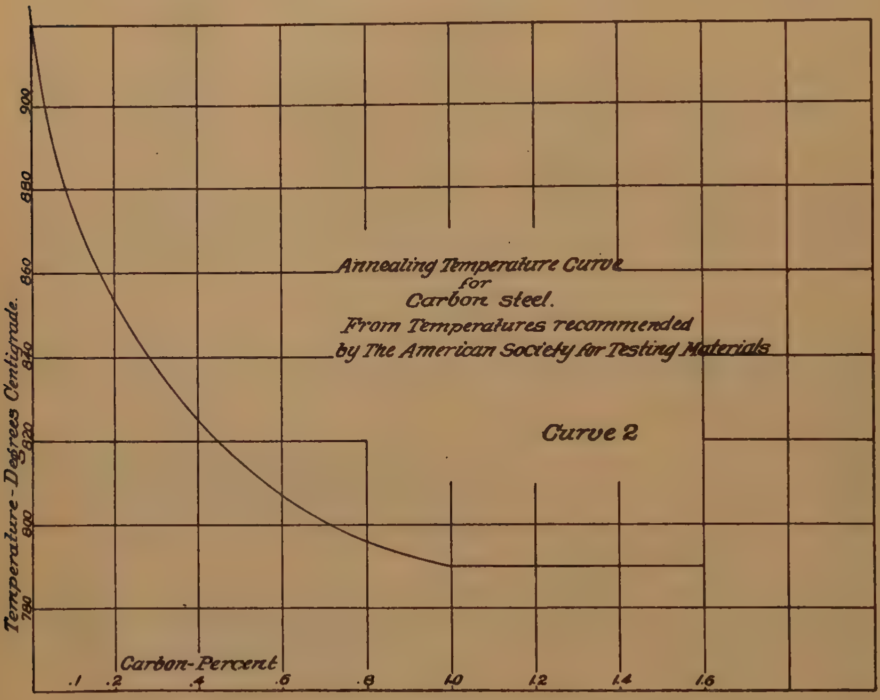


Fig. 2.

which is shown in Fig. 2. The proper heat-treating temperatures were then read from this curve.

The layout of heat treatments adopted was as follows: One tension specimen and four Charpy specimens were taken from each steel for each heat treatment. Specimens were tested: (a) As received, hot-rolled; (b) after annealing; (c) after hardening in water; (d) after hardening in oil; (e) after hardening in oil and drawing at 375°C.; (f) after hardening in oil and drawing at 460°C.; (g) after hardening in oil and drawing at 560°C.; (h) after hardening in oil and drawing at 650°C.

[1] *Year Book of the American Society for Testing Materials*, 1914, p. 201.

The annealing and hardening temperatures used for each steel were as follows:

Steel, Per Cent. Carbon	Annealing and Hardening Temperature, Degrees Centigrade
0.14	866
0.18	858
0.32	836
0.46	819
0.49	816
0.57	809
0.71	800
0.83	795
1.01	790
1.22	790
1.39	790
1.46	790

The steels were originally received as 12- or 14-ft. bars 1 in. in diameter. From these bars, tensile and Charpy specimens were rough-machined, and were left 1⁄16 in. large on all dimensions. Threads were not cut on the tensile specimens, nor were the Charpy bars notched. All specimens were to be finish-machined after heat treatment. All heatings were done in a small electrically heated closed muffle furnace, which maintained a very uniform temperature throughout the heating chamber. The inside dimensions of the muffle were 3¾ by 5⅜ by 11½ in.

The heating for annealing and hardening all specimens for each steel was done at one time. The procedure was as follows: The specimens were suitably arranged in the furnace when cold. A platinum-platinum rhodium thermocouple, the hot junction of which was inserted in a sample of the steel being heated, was also arranged in the furnace. The muffle door was closed with firebrick and asbestos, and the current turned on. It required approximately 1½ hr. for the specimens to come to heat. They were held at the desired temperature for 20 min. The specimens to be hardened were then quenched in oil or water as indicated; those to be annealed were allowed to cool slowly with the furnace, the opening to the muffle being again closed. Approximately 8 hr. was required for the furnace and annealed specimens to reach atmospheric temperature.

The drawing operation was performed as follows: A cast-iron pot, 12 by 12 by 12 in. inside dimensions, was filled with a eutectic mixture of sodium and potassium nitrates and was heated by gas. The melting point of this mixture is low enough so that the bath was molten at the desired temperatures. The bath was raised to the desired temperature and the specimens introduced in a wire basket. They were maintained at the drawing temperature for 30 min.

All temperature measurements were made by means of platinum-platinum rhodium thermocouples and a Leeds & Northrup precision

potentiometer. The thermocouples used were carefully calibrated before use.

Subsequent to heat treatment, all specimens were finish-machined

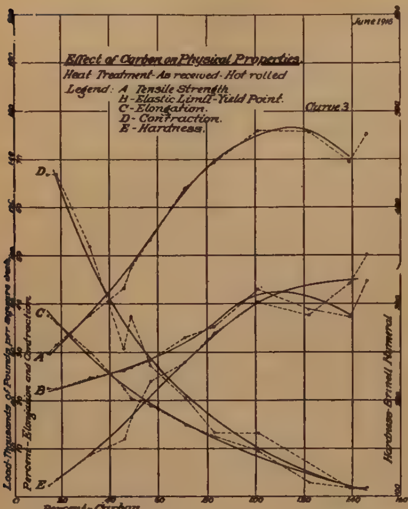


FIG. 3.

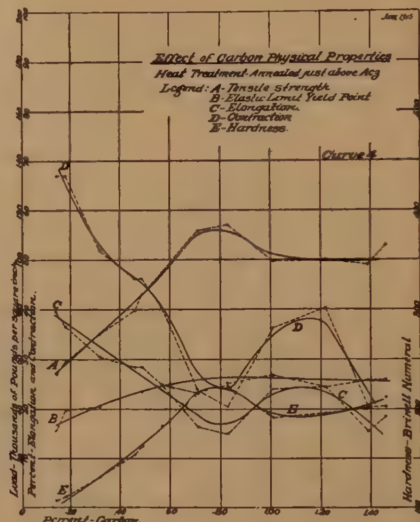


FIG. 4.

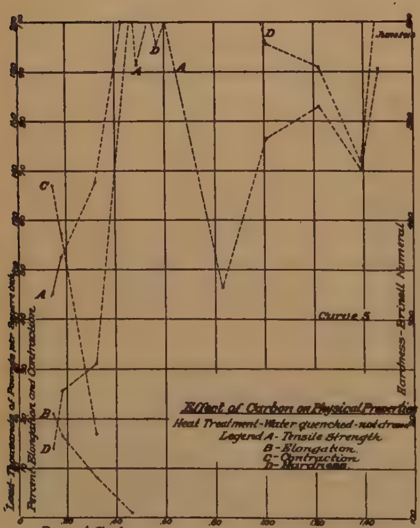


FIG. 5.

FIG. 6.

previous to testing. In the case of certain of the harder specimens, it was necessary to soften the ends so that a thread could be cut on them. This was accomplished, without affecting the reduced section of the

specimens, by partially immersing them in water and playing a flame on the end to be softened.

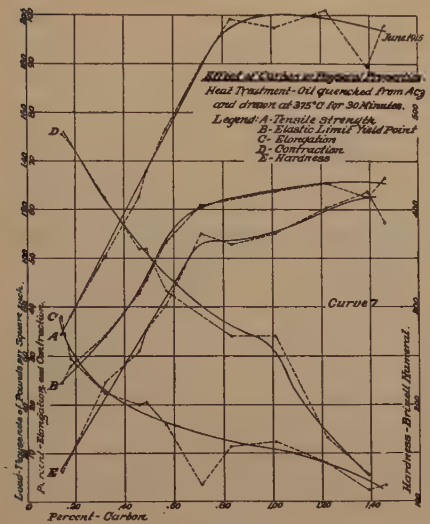


FIG. 7.

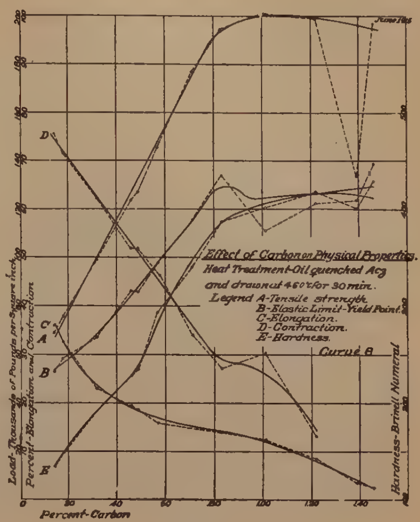


FIG. 8.

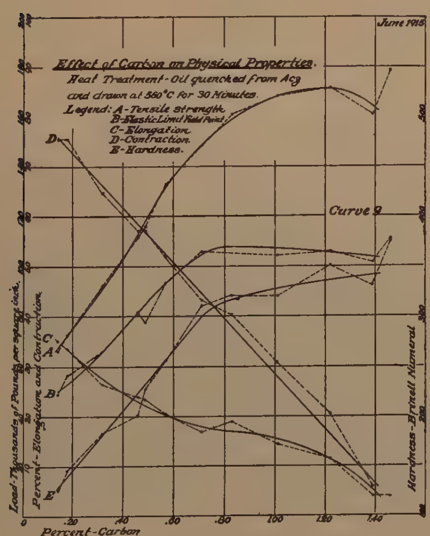


FIG. 9.

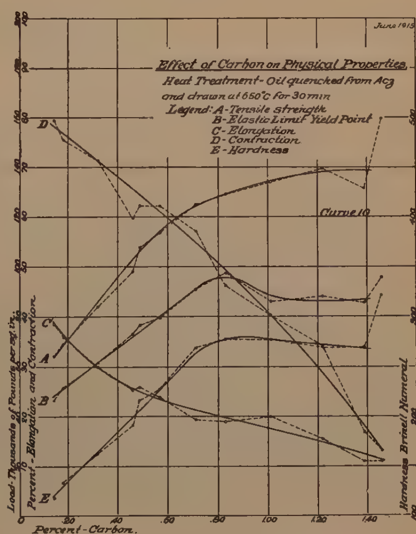


FIG. 10.

TESTING

All tensile tests were made on the 800,000-lb. Emery hydraulic-principle testing machine in the laboratory at Watertown Arsenal. Brinell hardness tests were made on an Alpha machine under standard

conditions; that is, using a 10-mm. ball, a load of 3,000 kg. and a time of application of 30 sec.

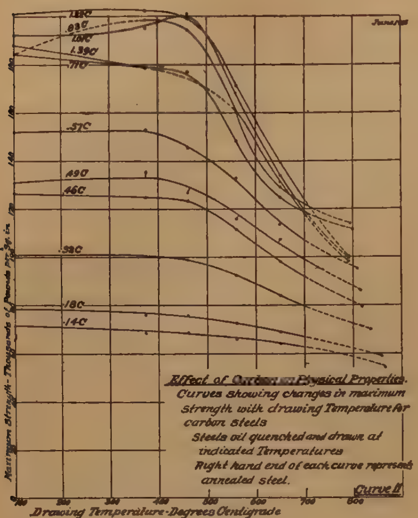


FIG. 11.

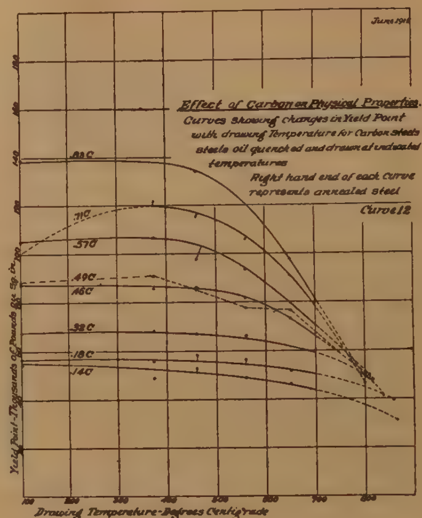


FIG. 12.

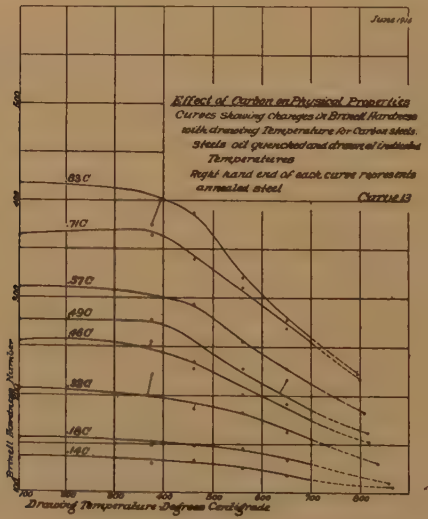


FIG. 13.

DATA

The results of the tensile and Brinell hardness tests are summarized in Tables II to IX; while Figs. 3 to 10 inclusive graphically depict these results, showing for each heat treatment the changes in physical

properties due to increase in carbon content. Figs. 11, 12 and 13 show the changes in maximum strength, yield point, and Brinell hardness, respectively, with increasing drawing temperatures. Curves for the steels with different carbon content are shown on the same sheet so that

FIG. 14.

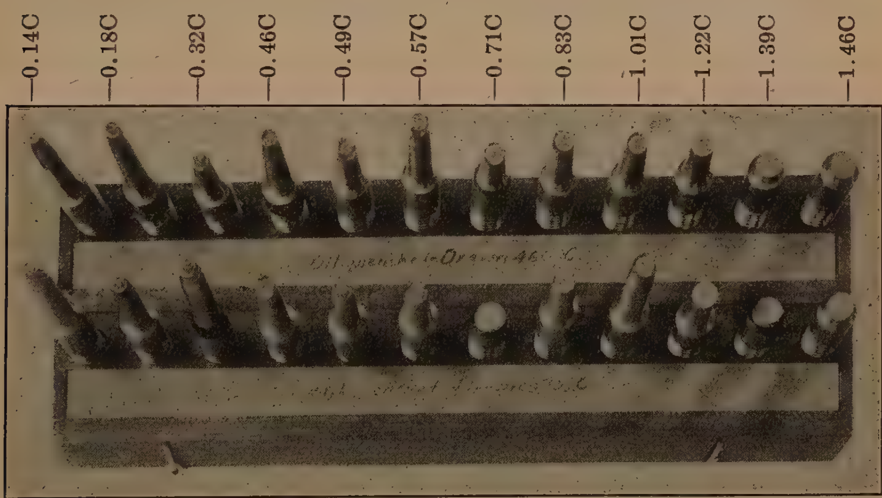


FIG. 15.

the relation between them is readily apparent. Photographs of the fractures of the tension-test specimens tested in this investigation are shown in Figs. 14 to 17. In all cases the specimens are arranged in the order of increasing carbon content from left to right, the heat treatments being indicated in the photographs.

Discussion

In plotting the curves in Figs. 3 to 10 inclusive, the observed points have been located and straight dotted lines drawn between them. Idealized curves have been drawn full following the general shape of the dotted curves. Fig. 5, showing results on the water-quenched speci-

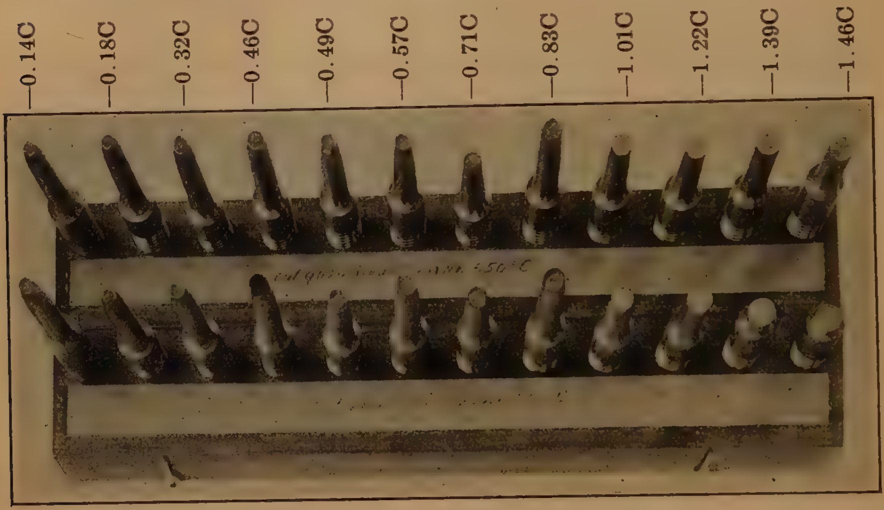


Fig. 16.

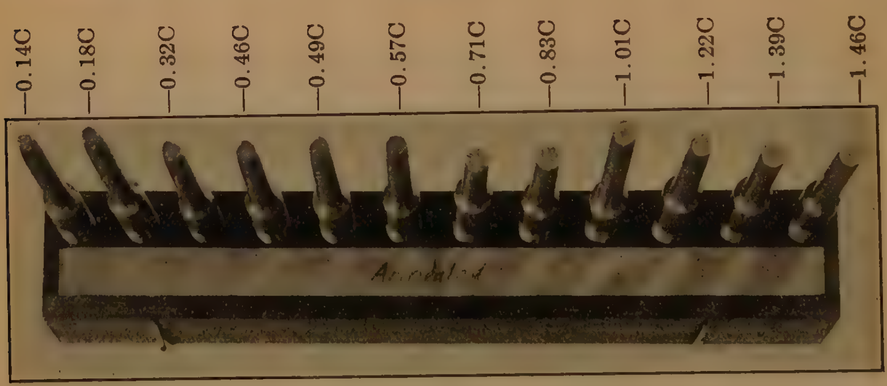


Fig. 17.

mens, is given for the sake of completeness. It is, however, of little value, for in the case of all steels examined with over 0.32 per cent. carbon, the unrelieved internal stresses doubtless caused irregular results to be obtained.

Maximum Strength.—In the annealed condition, Fig. 4, the maximum unit strength reaches a maximum value at eutectoid composition; i.e., 0.83 per cent. carbon, and then decreases slightly with increase of

carbon. With the heat-treated steels, however, Figs. 6, 7, 8, 9 and 10, and also in Fig. 3 for hot-rolled steels, the falling off in maximum strength does not begin until a composition of 1.20 per cent. carbon is reached. These curves show a rapid rise in tensile strength with increasing carbon to eutectoid composition, then a less rapid increase in strength to 1.20 per cent. carbon, followed generally by a slight falling off. The increase in strength in most cases at 1.46 per cent. carbon is to be attributed, at least partially, to the chromium content in this particular steel.

Yield Point.—In all cases, except the hot-rolled condition, Fig. 3, the unit stress of the yield point reached a maximum at eutectoid composition, falling off slightly or remaining constant with increase in carbon.

Brinell Hardness.—The curves for Brinell hardness are similar in shape in nearly all cases to the curves for maximum strength.

Elongation and Contraction.—In general, the curves representing these two qualities show decreasing values with increasing carbon. The tendency of the curves is to be concave upward—they are straighter, however, in the heat-treated steels than in the hot-rolled condition. In Figs. 4 and 6, that is with steels, in the annealed and the plain oil-quenched conditions, the curves show a gain in ductility just beyond the eutectoid composition which is unaccountable.

The curves in Figs. 11, 12 and 13 show the relation between the maximum strength, yield point, and Brinell hardness, respectively, and the drawing temperature after oil quenching for the various steels investigated. These curves show nicely the relative effects of increasing carbon in connection with hardening and quenching treatments. The final points on all the curves represent the annealed condition of the metal. The curves are, therefore, shown dotted between these last points and a drawing temperature of 700°C.

TABLE II.—*Specimens Tested, As Received*

Carbon, Per Cent.	Yield Point, Pounds per Square Inch	Ultimate Strength, Pounds per Square Inch	Elongation, Per Cent	Contraction of Area, Per Cent.	Elastic Ratio, Per Cent.	Brinell Hardness Number
0.14	45,000	59,500	37.5	67.0	75.6	112
0.18	45,000	63,000	36.0	67.0	73.0	118
0.32	49,500	75,500	30.0	51.9	65.5	144
0.46	52,500	86,500	22.5	30.7	60.7	160
0.49	54,000	95,000	20.5	37.2	56.8	183
0.57	57,000	106,500	19.0	27.4	53.5	220
0.71	66,000	128,000	15.0	20.5	51.5	240
0.83	70,500	139,000	12.5	13.3	50.7	269
1.01	86,000	152,000	9.5	13.3	56.5	302
1.22	77,500	151,500?	3.0?	....	51.1	288
1.39	74,500	139,000	2.0	1.8	53.6	321
1.46	89,500	150,500	1.5	1.8	59.4	351

Table III.—*Specimens Annealed*

Carbon, Per Cent.	Yield Point, Pounds per Square Inch	Ultimate Strength, Pounds per Square Inch	Elongation, Per Cent.	Contraction of Area, Per Cent.	Elastic Ratio, Per Cent.	Brinell Hardness Number
0.14	31,000	54,500	39.5	67.0	56.8	107
0.18	39,500	59,000	36.5	67.0	66.8	111
0.32	41,000	70,000	30.5	51.9	58.5	131
0.46	48,000	79,500	28.5	46.2	60.3	153
0.49	47,000	86,000	28.5	46.2	54.6	163
0.57	50,000	95,000	25.0	40.3	52.6	183
0.71	46,500	111,500	16.5	24.0	41.7	217
0.83	50,500	114,000	15.0	20.5	44.3	223
1.01	53,800	99,400	24.0	36.2	54.1	192
1.22	49,000	100,000	24.5	40.3	49.0	196
1.39	51,500	98,000	15.5	20.5	52.5	202
1.46	52,000	106,000	18.5	20.5	49.0	212

Table IV.—*Specimens Water Quenched*

Carbon, Per Cent.	Yield Point, Pounds per Square Inch	Ultimate Strength, Pounds per Square Inch	Elongation, Per Cent.	Contraction of Area, Per Cent.	Brinell Hardness Number
0.14	Not Determined.	90,000	21.0	67.0	170
0.18		105,000	16.5	57.2	228
0.32		135,500	8.0	16.9	255
0.46		220,000	1.0	0.0	600
0.49		183,000	0.0	0.0	713
0.57		215,000	0.0	0.0	578
0.71		....	....	....	....
0.83		93,000	0.0	0.0	744
1.01		153,000	0.0	0.0	578
1.22		166,000	0.0	1.8	555
1.39		140,500	1.0	0.0	460
1.46		181,500	0.0	0.0	627

Table V.—*Specimens Oil Quenched*

Carbon, Per Cent.	Yield Point, Pounds per Square Inch	Ultimate Strength, Pounds per Square Inch	Elongation, Per Cent.	Contraction of Area, Per Cent.	Elastic Ratio, Per Cent.	Brinell Hardness Number
0.14	56,500	71,500	34.0	75.5	79.0	134
0.18	54,500	78,700	31.0	75.2	69.2	156
0.32	67,500	101,000	23.5	62.3	66.8	207
0.46	87,500	126,500	20.5	51.9	69.1	255
0.49	88,500	131,000	17.5	51.9	67.5	277
0.57	105,000	152,000	16.5	40.3	69.0	311
0.71	100,000	184,500	1.5	0.0	54.2	364
0.83	138,500	184,500	4.0	5.7	75.0	418
1.01	127,000	192,500	10.5	34.0	65.9	387
1.22	122,500	201,500	9.0	16.9	60.8	402
1.39	97,500	188,500	2.5	5.7	51.7	430
1.46	122,500	201,500	3.5	5.7	60.7	418

TABLE VI.—*Specimens Oil Quenched and Drawn at 375°C.*

Carbon, Per Cent.	Yield Point, Pounds per Square Inch	Ultimate Strength, Pounds per Square Inch	Elongation, Per Cent.	Contraction of Area, Per Cent.	Elastic Ratio, Per Cent.	Brinell Hardness Number
0.14	49,000	68,500	38.0	75.5	71.5	131
0.18	56,000	76,000	29.5	73.5	73.6	149
0.32	68,000	100,500	22.5	62.3	67.0	223
0.46	85,500	125,000	20.0	51.9	68.4	255
0.49	91,000	135,500	20.5	51.9	67.1	277
0.57	106,500	153,000	16.0	43.3	68.8	302
0.71	121,400	179,100	3.5	....	67.8	375
0.83	....	198,500	11.5	34.0	....	364
1.01	127,500	195,000	12.5	34.0	65.3	375
1.22	131,000	203,000	8.0	13.3	64.5	402
1.39	125,000	178,500	2.5	5.7	70.0	418
1.46	133,000	196,000	3.5	....	68.8	387

TABLE VII.—*Specimens Oil Quenched and Drawn at 460°C.*

Carbon, Per Cent.	Yield Point, Pounds per Square Inch	Ultimate Strength, Pounds per Square Inch	Elongation, Per Cent.	Contraction of Area, Per Cent.	Elastic Ratio, Per Cent.	Brinell Hardness Number
0.14	53,000	68,500	36.0	75.5	77.3	134
0.18	58,500	75,500	32.5	71.4	77.4	149
0.32	67,000	99,000	23.0	62.3	67.6	187
0.46	86,000	123,500	19.5	51.9	69.6	228
0.49	85,500	127,000	18.5	51.9	67.3	235
0.57	97,500	145,500	16.0	46.2	64.4	293
0.71	115,500	177,000	10.0	34.0	65.2	340
0.83	134,000	194,500	14.0	37.2	68.8	387
1.01	111,000	201,500	12.5	40.3	55.0	402
1.22	122,500	199,000	8.5	13.3	61.5	418
1.39	123,500	134,000	3.5	....	91.4	402
1.46	139,000	197,000	2.5	....	70.5	430

TABLE VIII.—*Specimens Oil Quenched and Drawn at 560°C.*

Carbon, Per Cent.	Yield Point, Pounds per Square Inch	Ultimate Strength, Pounds per Square Inch	Elongation, Per Cent.	Contraction of Area, Per Cent.	Elastic Ratio, Per Cent.	Brinell Hardness Number
0.14	49,000	66,000	35.0	75.5	74.2	126
0.18	56,500	72,500	33.5	75.5	77.9	146
0.32	66,000	92,500	26.5	64.7	71.3	183
0.46	81,500	111,500	24.0	57.2	73.0	202
0.49	77,500	116,000	23.5	57.2	66.8	228
0.57	93,500	133,000	20.5	51.9	70.3	255
0.71	106,000	148,500	17.0	43.3	71.3	311
0.83	....	161,000	19.0	40.3	....	321
1.01	104,500	168,500	14.5	30.7	62.0	321
1.22	106,000	171,500	11.5	20.5	61.8	351
1.39	101,500	161,000	4.0	5.7	65.5	332
1.46	111,000	178,500	4.0	....	62.2	375

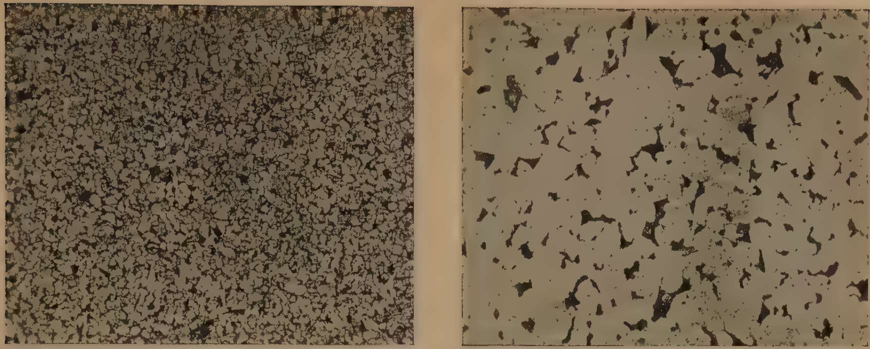
TABLE IX.—*Specimens Oil Quenched and Drawn at 650°C.*

Carbon, Per Cent.	Yield Point, Pounds per Square Inch	Ultimate Strength, Pounds per Square Inch	Elongation, Per Cent.	Contraction of Area, Per Cent.	Elastic Ratio, Per Cent.	Brinell Hardness Number
0.14	46,500	64,000	38.5	79.5	72.6	118
0.18	51,500	69,000	36.0	75.5	74.6	134
0.32	61,500	84,000	30.0	71.4	73.2	163
0.46	73,000	98,000	25.5	59.8	74.4	192
0.49	76,500	107,500	26.0	62.3	71.1	217
0.57	79,500	113,500	24.0	62.3	70.0	228
0.71	91,000	125,000	19.5	57.2	72.7	269
0.83	97,500	129,000	19.0	46.2	75.5	277
1.01	86,000	134,000	20.0	40.3	64.1	277
1.22	88,000	139,500	15.5	34.0	63.1	269
1.39	85,500	131,500	11.0	16.9	65.0	269
1.46	95,500	159,500	11.0	13.3	59.2	321

APPENDIX

The accompanying photomicrographs (Figs. 18 to 29) are presented as being of considerable interest in connection with the preceding experiments. The 12 steels investigated are shown in two different conditions: (1) as received, hot-rolled and (2) normalized or annealed from 1,000°C. In the normalized series the effect of carbon on the microstructure is most readily apparent. The micrographs of the hot-rolled steels appear on the left-hand side of the illustrations; those of the normalized steels on the right-hand side.

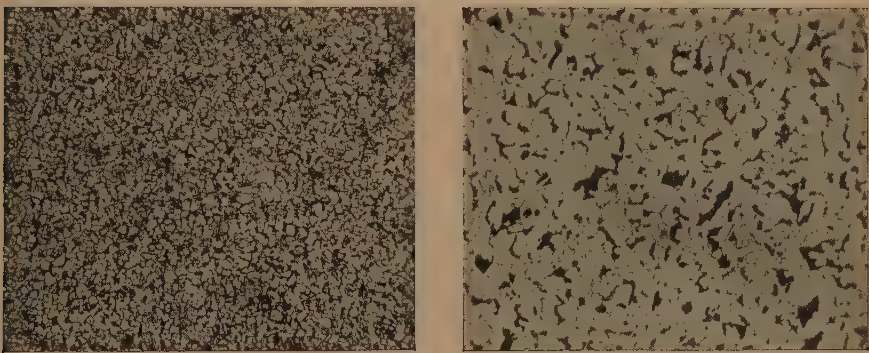
In the case of the hot-rolled steels, the structures consist of sorbito-pearlite and ferrite for the hypo-eutectoid steels, and sorbito-pearlite and cementite for the hyper-eutectoid steels. In the case of the normalized steels, the grain structures are considerably larger and the constituents are pearlite and ferrite for hypo-eutectoid steels, and pearlite and cementite for hyper-eutectoid steels. All micrographs are magnified 51 diameters.



As received. Hot rolled. Brinell hardness, 112.

Normalized. Annealed from 1,000°C. Brinell hardness, 107.

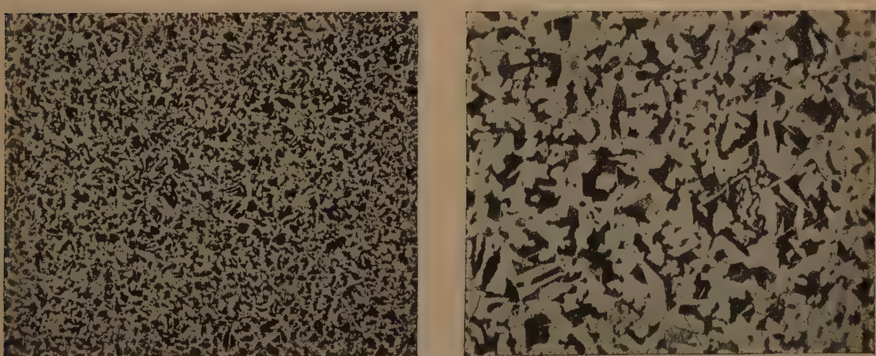
FIG. 18.—ANALYSIS: C, 0.14; Mn, 0.45; Si, 0.131; S, 0.035; P, 0.018 Per Cent.



As received. Hot rolled. Brinell hardness, 118.

Normalized. Annealed from 1,000°C. Brinell hardness, 111.

FIG. 19.—ANALYSIS: C, 0.18; Mn, 0.56; Si, 0.132; S, 0.043; P, 0.024 Per Cent.



As received. Hot rolled. Brinell hardness, 144.

Normalized. Annealed from 1,000°C. Brinell hardness, 134.

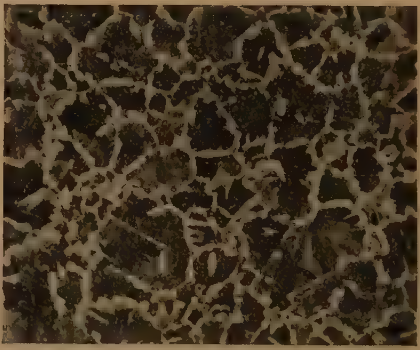
FIG. 20.—ANALYSIS: C, 0.32; Mn, 0.51; Si, 0.128; S, 0.027; P, 0.009 Per Cent.

As received. Hot rolled. Brinell hardness, 160.

Normalized. Annealed from 1,000°C. Brinell hardness, 155.

FIG. 21.—ANALYSIS: C, 0.46; Mn, 0.40; Si, 0.144, S, 0.50; P, 0.020 Per Cent.

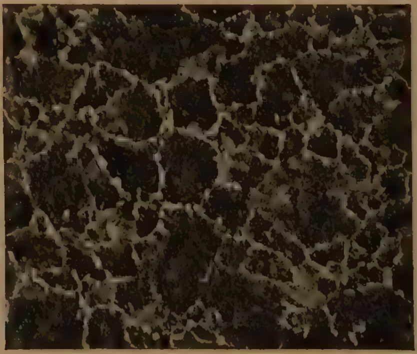
As received. Hot rolled. Brinell hardness, 183.



Normalized. Annealed from 1,000°C. Brinell hardness, 179.

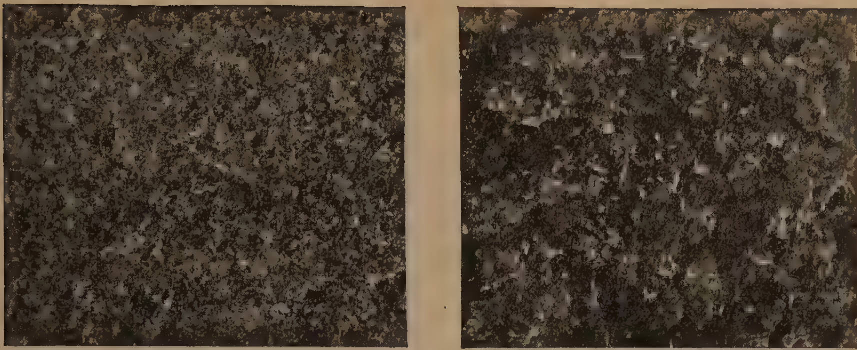
FIG. 22.—ANALYSIS: C, 0.49; Mn, 0.60; Si, 0.127; S, 0.028; P, 0.013 Per Cent.

As received. Hot rolled. Brinell hardness, 220.



Normalized. Annealed from 1,000°C. Brinell hardness, 206.

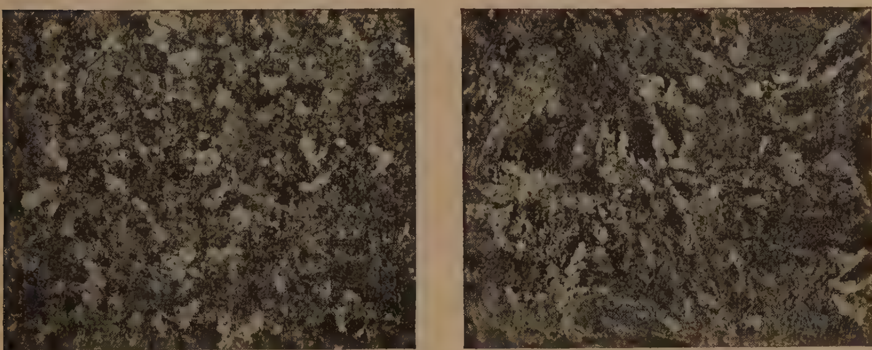
FIG. 23.—ANALYSIS: C, 0.57; Mn, 0.65; Si, 0.167; S, 0.028; P, 0.012 Per Cent.



As received. Hot rolled. Brinell hardness, 240.

Normalized. Annealed from 1,000°C. Brinell hardness, 235.

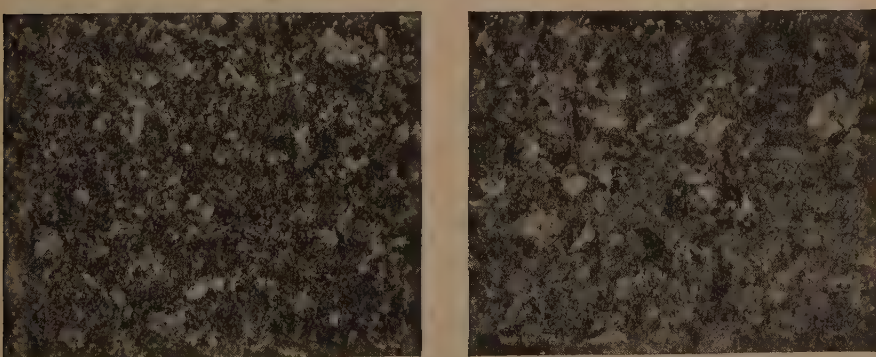
FIG. 24.—ANALYSIS: C, 0.71; Mn, 0.67; Si, 0.147; S, 0.035; P, 0.027 Per Cent.



As received. Hot rolled. Brinell hardness, 269.

Normalized. Annealed from 1,000°C. Brinell hardness, 257.

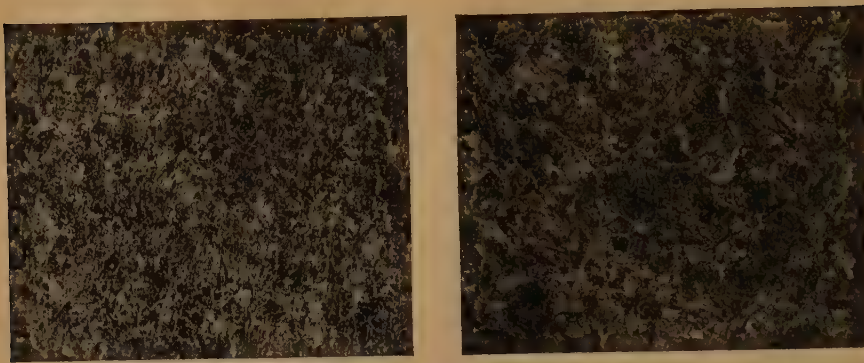
FIG. 25.—ANALYSIS: C, 0.83; Mn, 0.55; Si, 0.152; S, 0.028; P, 0.018 Per Cent.



As received. Hot rolled. Brinell hardness, 302.

Normalized. Annealed from 1,000°C. Brinell hardness, 255.

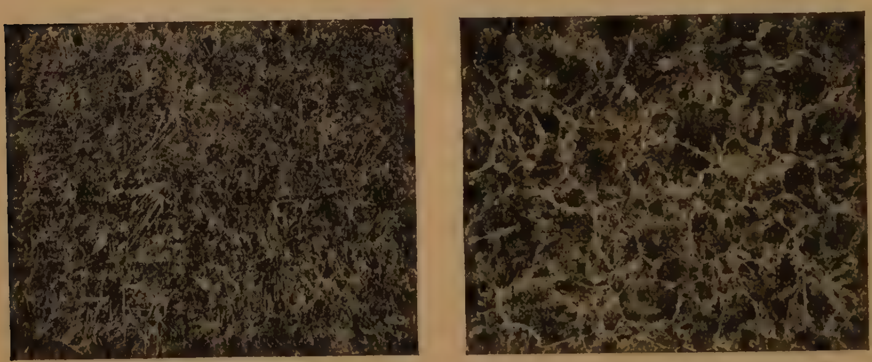
FIG. 26.—ANALYSIS: C, 1.01; Mn, 0.39; Si, 0.160; S, 0.029: P, 0.016 Per Cent.



As received. Hot rolled. Brinell hardness, 288.

Normalized. Annealed from 1,000°C. Brinell hardness, 269.

FIG. 27.—ANALYSIS: C, 1.22; Mn, 0.34; Si, 0.181; S, 0.031; P, 0.025 Per Cent.



As received. Hot rolled. Brinell hardness, 321.

Normalized. Annealed from 1,000°C. Brinell hardness, 255.

FIG. 28.—ANALYSIS: C, 1.39; Mn, 0.20; Si, 0.191; S, 0.029; P, 0.015 Per Cent.

As received. Hot rolled. Brinell hardness, 351.

Normalized. Annealed from 1,000°C. Brinell hardness, 302.

FIG. 29.—ANALYSIS: C, 1.46; Mn, 0.20; Si, 0.133; S, 0.035; Cr, 0.35; P, 0.011 Per Cent.

Discussion

E. D. Campbell, Ann Arbor, Mich.—Any one at all familar with research work can at once see that Mr. Nead has condensed into his tables and curves the results of a carefully planned set of experiments which must have involved a great amount of work on his part. He deserves much credit for the way in which he has carried out these experiments, for it is to the results of such systematic experimentation that we must look for further knowledge of the underlying causes which determine the physical properties of steel. It will probably be many years before we have sufficient knowledge of all the factors of the problem of the constitution of steel, but every set of carefully carried out experiments such as Mr. Nead has made adds something to our knowledge and brings us a little nearer to the final solution of the problem.

For many years there have been differences among metallurgists concerning the underlying causes of the changes in physical properties brought about by different heat-treatments of steels. If we hope ever to arrive at a satisfactory explanation of the relations existing between chemical composition, heat-treatment, and physical properties of steels, it will be necessary to formulate a definite working hypothesis, which may serve as a working basis for carrying on experimental work calculated to prove or disprove the conceptions involved in the working hypothesis. Most metallurgists are now agreed that the element iron is capable of existing in at least two molecular states; first, the soft, or, as it is usually called, alpha form, characteristic of very slowly cooled metal, and at least one allotropic form produced by raising the temperature above the A_3 point. Whether there is more than one allotropic form of iron or whether the molecular change of the metal when heated above the critical range is or is not, in reality, an allotropic one, is still a subject of discussion among metallurgical chemists. In this laboratory the assumption is made that when iron in the alpha form is heated above the critical range it undergoes an allotropic change. J. E. Stead has demonstrated that the transformation of pure iron from the allotropic to the alpha form on cooling takes place with great rapidity, probably requiring only a fraction of a second for completion, unless the presence of some element other than iron retards the rate of transformation. A further assumption is made that iron in the allotropic form is capable of holding in solid solution a much higher concentration of solutes, whether these solutes be carbides, phosphides, or other compounds of iron, than can be held by iron in the alpha form, since it has been well known for many years that when steel is annealed and the iron thus converted to the alpha form, almost all of the carbides, and frequently other solutes, separate out from solid solution so that they may be easily recognized under the microscope.

Ever since the classical work of Abel about 30 years ago, when he showed by the analysis of pure carbide of iron recovered from annealed steel, that the empirical formula Fe_3C might be used to express the composition, this formula has been employed by many metallographists not only to represent pure carbide of iron but also the metallographic constituent cementite. We now have almost conclusive evidence[1] to show that a formula so simple as Fe_3C involving a single carbon atom to the molecule cannot be used to express the molecular composition of even pure carbide of iron, but that a multiple of this formula $(Fe_3C)_n$ is the simplest empirical formula capable of representing pure carbide of iron. We have reason to believe that the element carbon, which is capable of forming that large number of complex hydrocarbons which constitute petroleum, does not lose its most striking property simply because the hydrogen of the hydrocarbons is replaced by iron or some other metal. If cementite used in a broad generic sense, like petroleum, is regarded as being composed of a mixture of carbides or ferrocarbons whose molecular composition is dependent upon the carbon concentration, heat-treatment, and presence or absence of metals other than iron, we might develop a conception of the chemical constitution of any given steel under given thermal conditions that would be more nearly in accord with known facts than is possible under any assumption that the composition of cementite is really represented by the formula Fe_3C. It is the simplicity of this latter formula that appeals so strongly to many metallographists and makes them unwilling to give up the old assumption, even though it does not harmonize with known facts. If iron be considered a solvent and the chemical constitution of steel be regarded as a study in solubilities, we would then find, as in ordinary aqueous solutions, that the physical properties of the solvent are most profoundly modified by the nature and concentration of those substances which are in solution, while substances which are merely mechanically mixed or in suspension exert comparatively little influence. Substances which are in excess of what can be held in solution are often deleterious since they tend to destroy the homogeneity of the mass.

Although most of the properties of steel such as yield point, ultimate strength, elongation and contraction of area, depend almost entirely on the chemical constitution, the hardness as measured by the Brinnell method is apparently a somewhat more complex phenomena. The Brinnell Testing Machine is, in reality, a viscometer for measuring the flow of a solution of high viscosity. There seem to be two kinds of hardness, described by Hadfield as brittle and tough hardness. The

[1] E. D. Campbell: A Pure Carbide of Iron, *American Chemical Journal*, vol. xviii, pp. 836 to 847 (December, 1896); also The Constitution of Steel, *Journal of the Iron and Steel Institute*, vol lvi, pp. 223 to 233 (1899); also The Constitution of Carbon Steels, *Journal of the Iron and Steel Institute*, vol. lxxviii, pp. 318 to 335 (1908).

extreme hardness of over 700 Brinnell units, characteristic of water-quenched carbon steels, has been variously attributed by Metcalf to a state of internal tension, by Le Chatelier to the existence of alpha iron in a state of internal strain, and recently by McCance to the retention of a certain portion of iron trapped during its transformation from the allotropic to the alpha form and retained in a state of interstrain. It is apparently the existence of a certain portion of the metal in a state of strain which increases so greatly the resistance to flow under the pressure of a Brinnell Ball, but this same state of strain which results in extreme hardness or resistance to flow also results in brittleness. On the other hand a very considerable hardness is due to the concentration of carbides retained in solid solution, when the rate of cooling of the metal while passing through the transformation range is slow enough to permit almost complete transformation of the iron without retention in the interstrained state, but not so slow as to give time enough for precipitation from solid solution of much of the dissolved carbides. The difference between the hardness due to these two causes is illustrated by the well-known difference in behavior of water-quenched and oil-hardened steel.

Metallography of Steel for United States Naval Ordnance

The Requirements of the Bureau of Ordnance

BY HAROLD EARLE COOK,* PHILADELPHIA, PA.

(New York Meeting, February, 1916)

The purpose of this paper is to state briefly the inspection requirements of the Bureau of Ordnance, the specifications governing the inspection, and the physical and chemical properties of the steel used in the construction of ordnance for our Navy.

It would be of interest to evoke discussion on points in these specifications which do not tend to insure the production of the very best material for the purpose intended. In other words, what requirements of our specifications are useless and also in what manner are our specifications deficient?

It might be well at the start to answer the first criticism of this paper which will probably be made, and that is that the title does not agree with the subject matter.

Considering metallographic work as covering only the macroscopic and microscopic examination of metals, this paper will have gone beyond the subject; but it is believed that a broader definition of metallography is that it is the study of the relation of the internal structure of metals and alloys to their composition and to their physical and other properties.

The internal structure of steel is dependent on its physical and its chemical properties. These physical properties are dependent on the chemical composition and the mechanical and thermal treatment of the steel. So it seems that this subtitle would not be entirely necessary and that the title "Metallography of Steel for Ordnance Purposes" would cover a study, not only of what we can learn of macroscopic and microscopic examination, but broadly of the whole subject of the specifications, the fulfillment of the requirements of which produce the structural conditions revealed by macroscopic and microscopic examination.

I will refer but briefly at the beginning to the microscope and its uses in our inspection. It is only very recently that the Bureau of Ordnance has furnished certain of its inspection offices with a portable Tassin outfit for the microscopic examination of steel. Since that

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* Lieutenant-Commander, U. S. Navy, Naval Inspector of Ordnance, Works Midvale Steel Co.

time, the offices thus supplied have been studying the subject, and making considerable progress in order to keep up with the development of the art, so that when it reaches the state of development whereby it may be used to definite purpose in inspection, the various inspection offices will be equipped and prepared.

With the exception of our specifications for gun forgings there are no ordnance specifications at the present time which make reference to metallographic examination, as it may be said that defects which might be developed by routine microscopic examination would certainly be developed in the course of ordinary tests and inspection. Our specifications for gun forgings say: "The Department reserves the right to make such microscopic examination as it may deem necessary." The use of the microscope will be referred to in discussing the various specifications.

A few years ago each of the Bureaus under the Navy Department had its own specifications for the different classes of material required by it. For instance, as to steel forgings, the Bureau of Construction and Repair had its own specifications for material for forgings for hull material; the Bureau of Steam Engineering had its own specifications for engine forgings; and the Bureau of Ordnance had its own specifications covering forgings for mount work and mechanisms and gun forgings. Each Bureau had its own specifications for nonferrous alloys. This led to considerable trouble and complications with the various manufacturers; and recently the Navy Department has been endeavoring to combine, as far as possible, the specifications of the various Bureaus, getting them out as Navy Department specifications in leaflet form. A booklet of General Specifications is issued giving the requirements common to all material, method of testing and instructions to inspectors, which forms a part of each individual specification. These specifications are now used by the Bureau of Ordnance for ordinary forgings and castings, both of steel and nonferrous material for use in mount work, small parts of breech mechanisms and miscellaneous ordnance work. But for the more important ordnance material it is still necessary to issue separate and detailed specifications. Thus, there are separate detailed specifications for armor, projectiles, gun forgings, and torpedo air flasks and heads.

The specifications for armor are issued in booklet form prior to the award of each armor contract. The details of these specifications are confidential. The manufacturer is required to furnish the Bureau's inspector with reports of chemical analyses, etc., and the required amount of discard and forging reduction, as specified. The armor for each battleship is divided into groups of from 500 to 600 tons, which means 10 to 20 plates, depending on their size and thickness. As each armor plate in a group receives its final treatment, physical-test specimens are taken to show the properties of the plate, analyses are furnished, and other physical data; and when the last plate of a group receives its treat-

ment, one of the plates is selected for ballistic test. This plate is sent to the Naval Proving Ground at Indian Head, Md., and subjected to a ballistic test, being required to withstand the impact of a certain number of projectiles at given velocities.

The microscope plays no part in the inspection of armor, but this office has been making examination for purposes of study, of fragments of armor plate, and it is believed that some day information gained by the use of the microscope may prove of great value, at least in the manufacture of armor.

Specifications for the inspection of armor-piercing projectiles are also held confidential. As each lot of armor-piercing projectiles, when completed, is subjected to a ballistic test, a certain number of projectiles being selected from the lot for this purpose, great latitude is allowed the manufacturer as to the method of manufacture, discard, composition, etc.; but to promote uniformity in the lot, a maximum and minimum variation from the average of each of the important alloys used is prescribed. Physical tests are prescribed for the material used in the base plug of the shell, and also for the copper rotating band. Though it is believed that metallographic examination is of the greatest value to the manufacturer of armor-piercing projectiles, it is not thought that it will ever be of any practical value for inspection purposes, as the ballistic tests are such as to insure the excellence of a lot of projectiles which successfully endures the test.

In the manufacture of both armor and projectiles, every operation is open to the inspector, who is a commissioned officer.

Navy specifications for steel castings give requirements for six different grades. Of these, two grades in particular are used for ordnance material.

Grade "F," which is used for castings for gun yokes and special castings, must show a tensile strength of 85,000 lb. per square inch, a yield point of 53,000 lb. per square inch, with 22 per cent. elongation in 2 in., and 30 per cent. reduction of area. A bending test must give a 120° bend about a 1-in. diameter. It is required that this grade contain from 3.25 to 3.75 per cent. of nickel and that the phosphorus and sulphur content be not over 0.05 per cent.

Grade "B" is used for most ordinary castings and for those of great size. Its physical and chemical requirements are a maximum tensile strength of 80,000 and a minimum of 60,000 lb., with a yield point equal to 45 per cent. of the tensile strength obtained; elongation in 2 in., 22 per cent. and reduction of area 25 per cent. The same bending test is required as for Grade "F," and the phosphorus must not be over 0.06 nor the sulphur in excess of 0.05 per cent. The maximum and minimum tensile-test requirements are to insure proper machineability.

Castings for ordnance purposes vary from those of the very smallest size to massive castings, such, for example, as a slide for a 14-in. gun

mount, in which the complete casting, before machining, weighs in the neighborhood of 17 tons.

The specifications require that coupons from which test specimens are to be taken shall, whenever practicable, be cast on the body of the casting, and that, when this is not practicable, the coupon shall be cast with, or gated to, the casting, or with runners to the gate. The inspector's approval must be obtained in all cases, and the number of tests taken must be such as to exhibit thoroughly the uniform quality of the casting.

It has been my experience that tests taken from coupons gated to a casting are not indicative of the true condition of the body of the casting. The solidification of the metal may not be under the same conditions. The coupon would freeze more rapidly than the greater mass of metal. The segregation of the constituents would not be the same. The treatment effect might be quite different, especially if the requirements laid down for treatment in specifications, which will be discussed later, were permitted to be departed from. I believe that it would be much more desirable, where possible, to machine the test specimens from desired points in the body of the casting, but of course this procedure would be objected to strongly as increasing the expense.

With castings so small as to render it impracticable to cast coupons on the body of the casting, and where a number are made from the same heat and treated together, I believe that the preferable method of test is to require a sufficient extra number of castings to be made from each heat to select at random a number to be cut up for test. To cast, as is sometimes done, a coupon for test separate from the intricate-shaped castings, and to test it to pass upon a lot, or to test representatively castings differing in shape and size, gives to my mind, a false indication of the condition of the castings themselves.

In the larger castings it is the custom to cast coupons for test at various points on the cope and drag side of the casting, about 2 by 4 by 8-in. in size. On the castings as supplied to the Naval Gun Factory, coupons of this size form a portion of the pattern. These are cast so that the 4 by 8-in. area is fixed to the body of the casting, and they are used if at any time after delivery it be considered advisable to take additional tests from the castings.

As to the relative value of tests from coupons, I will cite an example. A manufacturer of some large, heavy castings, used coupons of approximately the size above named, but with the narrow edge of the coupon continuous to the body of the casting, so that the coupon could be easily knocked off, avoiding machining. Test specimens from these coupons gave satisfactory physical values and the castings were provisionally accepted. After delivery at the Gun Factory, additional tests were taken from some flat coupons which were machined off the castings; and so marked a difference was noted, both in all the physical properties and in

the microstructure as to preclude the use of these castings for the purpose intended. A number of them were re-treated and subsequently accepted on tests taken out of the body of the casting as furnished, no more coupon remaining. It was with great difficulty that the requirements were met.

An examination of the microstructure exhibited in the various test bars showed a wide variation in structure. The structures were nearly all pearlitic, but those from the original coupons showed a grain refinement not exhibited in the body of the casting at that time; and the pearlite was in a different state. These castings had been treated in a rather crowded casting pit, and it is thought that the coupons on edge projecting out from the body were more affected by the heat and that their rate of both heating and cooling differed from that of the rest of the casting. And further, it is possible that the loss of heat by radiation from the exposed surfaces of the coupon, exceeded the rate at which it could receive heat by conduction in cooling from the body of the casting. It is also probable that the furnace heat was insufficient to cause the body of the casting to reach the desired temperature for proper treatment.

In castings of which the shape and size would permit air- or oil-quenching, it is believed that the difference which I have noted would be very much more marked.

The specifications include a clause on treatment, in which it is specified that castings to be annealed shall be held at the temperature of annealing long enough to insure that the interiors of the castings have been brought to that temperature. After the castings have been soaked at the proper annealing temperature, they must be allowed to cool slowly in the furnace, carefully protected from drafts or air. Unless otherwise directed by the inspector, castings must not be removed from the furnace until they have been cooled down to the temperature at which the color dies. Under no circumstances will the manufacturer be allowed to use water, brine, oil or air-blast to hasten the cooling process, without the approval of the inspector. The number of hours requisite for raising the castings to the proper temperature, the length of time during which they should be soaked at that temperature, and the period required for slow cooling in the furnace, or in the air, may be prescribed by the Bureau concerned if it so desires.

Large castings are generally considered a cheap product in comparison with forgings, and appear never to have received the consideration as to scientific treatment and endeavor to give them a good structure that is bestowed upon high-grade forgings. It appears to be only quite recently that casting plants have begun to install pyrometric control for regulation of temperature. It appears to be still a disputed question whether a casting could ever be put in as good a condition as a forging. It is my belief that the most scientific treatment would fail to render a casting the equal of a forging as carefully made and treated. I believe that the

primary ingot crystallization can be completely destroyed only by means of mechanical working, or, if this theory is not correct, that the different structural conditions developed in the body of a massive casting in the ingot state, cannot be brought to a uniform condition by thermal treatment.

I am conducting, in a small way, some experiments with specimens of castings. I was first directed to this opinion by the fact that in numerous specimens examined microscopically, broad ferrite bands were noted from time to time extending across the field of the microscope, which did not appear to be in the nature of the usual "ghost lines" formed. It has occurred to me that the primary crystallization of the ingot which has crystals so large that they are often noted by the eye without magnification, or noted in macroscopic examination, are never completely destroyed. Heat treatment may dissolve the ferrite from the boundaries to such an extent that they are completely masked and cannot be seen by the microscope, or even if they are, the primary crystals may be so large that the usual field of magnification is contained in one large cell. This theory, of course, is not a new one, and has been the subject of discussion, the majority of authorities appearing to disbelieve it.

There are certain points open to discussion, which I will touch upon, but inasmuch as the treatment is largely left to the approval of the inspector, when a different logical treatment is desired for physical properties, his approval could be obtained. For instance, the temperature of 1,500° prescribed would manifestly give poor results if but a single treatment were attempted, as prescribed, with a grade "F" casting containing a rather high percentage of nickel and possibly of carbon, since long exposure at a temperature considerably above the upper critical would result in a large grain size. Many of the castings, specimens of which I have examined, show clearly the effect of this treatment.

In some cases large castings are placed in pits, heated up to a somewhat uncertain temperature, and held (at times) for as long as a day, and then cooled slowly in the furnace for an equal or longer time. The long exposure at high temperature is considered necessary to insure the soaking of the casting, but it certainly also insures a grain growth (provided the critical temperature has been exceeded). The extremely slow cooling through the critical range allows the rejection in totality of the ferrite, and in the low-carbon castings it often resolves itself into extremely large ferrite grains with large isolated pearlite patches in the grain boundaries.

It seems remarkable that physical tests from such castings give the results that they do, generally meeting the requirements. It is believed that impact or vibratory tests on this material would develop deficiencies in comparison with more scientifically treated material.

Some castings which have received a double annealing treatment

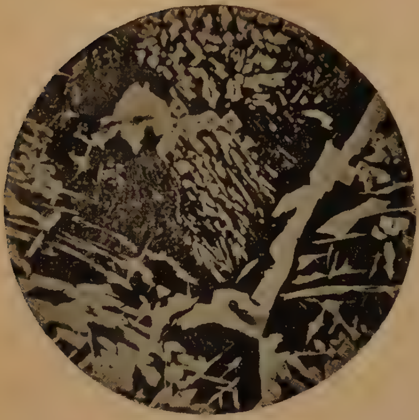

Fig. A. Fig. B.
Ingot steel, as cast. Carbon, 0.28 per cent. × 68.

Fig. C.—Ingot steel, as cast. × 21.

Fig. D.—3.5 per cent. nickel steel, as cast. × 68.

Plate I.

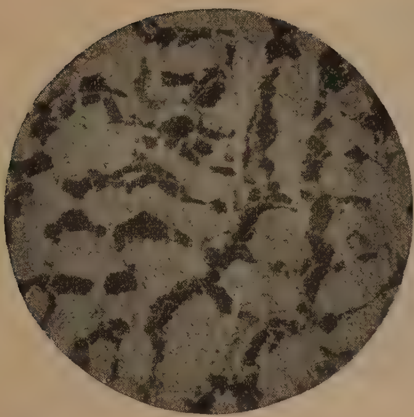

FIG. A.—× 80.

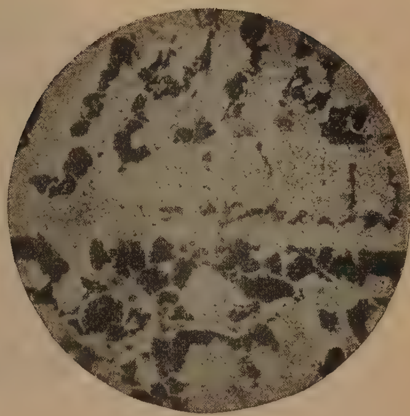

FIG. B.—× 80.

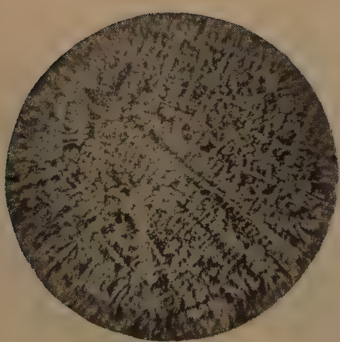

FIG. D.—× 21.

FIG. C.—× 21.

PLATE II.—STEEL CASTING.

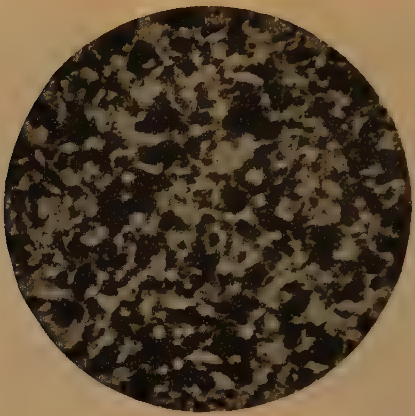

FIG. A.—× 80.

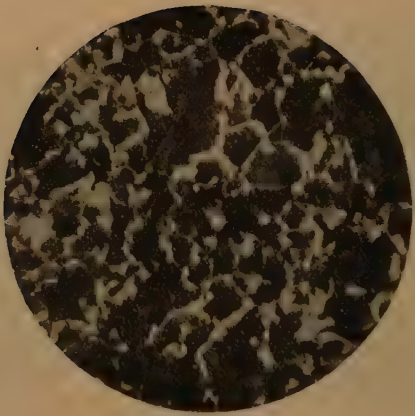

FIG. B.—× 80.

PLATE III.—NICKEL-STEEL CASTINGS.

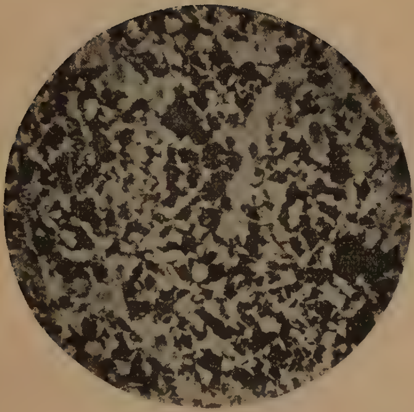

FIG. A.—× 80.

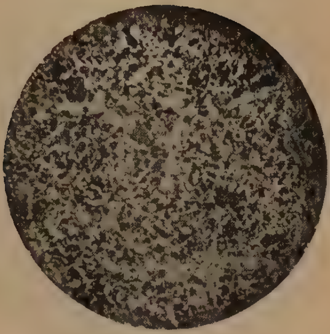

FIG. B.—× 32.

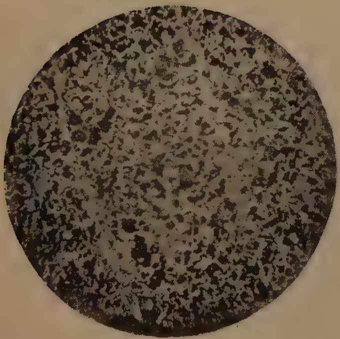

FIG. C.—× 32.

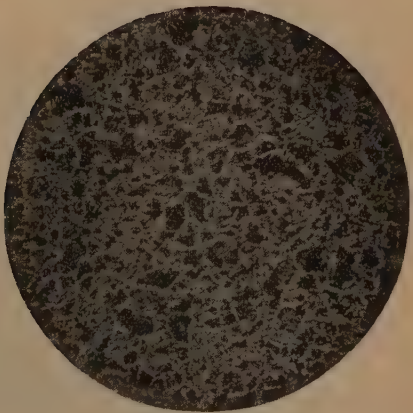

FIG. D.—× 80.

PLATE IV.—STEEL CASTINGS.

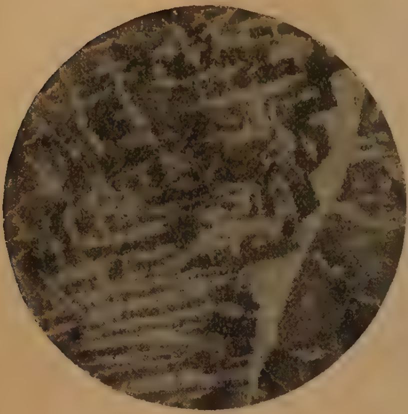

Fig. A.—× 80.

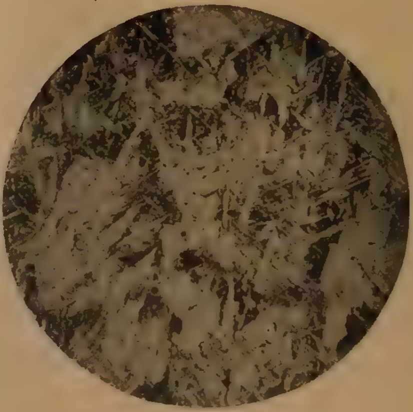

Fig. B.—× 80.

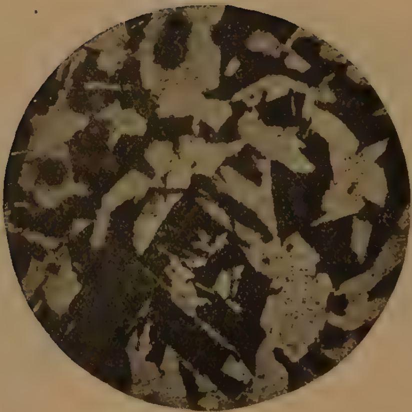

Fig. C.—× 80.

Plate V.

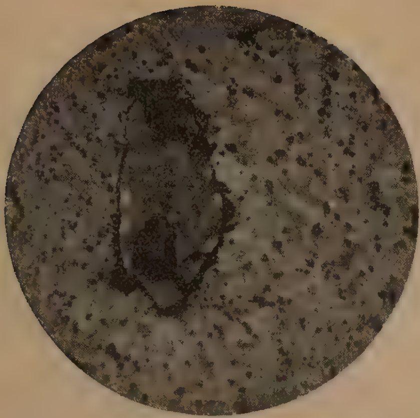

FIG. A.—0.30 per cent. carbon. × 80.

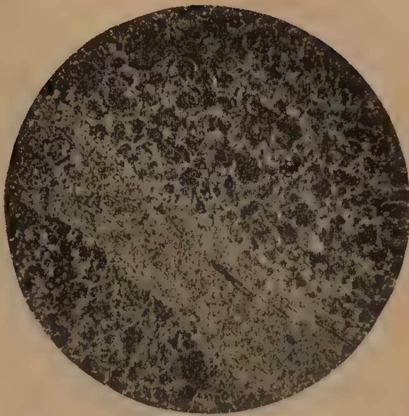

FIG. B.—0.40 per cent. carbon. × 80.

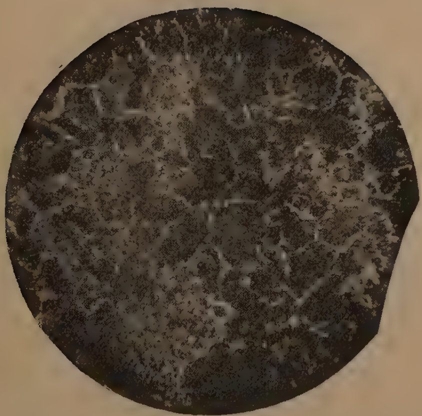

FIG. C.—0.42 per cent. carbon.

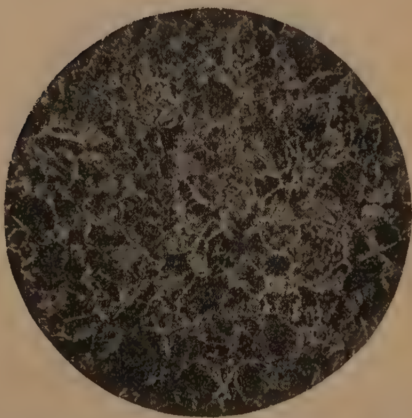

FIG. D.—Carbon, 0.40 per cent; nickel, 2 per cent. Annealed, quenched, drawn. Tensile strength, 96,500 lb. per square inch.; elastic limit 66,000 lb. per square inch; elongation in 2 in., 22.5; 59.9 per cent. reduction in area.

PLATE VI.—NICKEL STEEL.

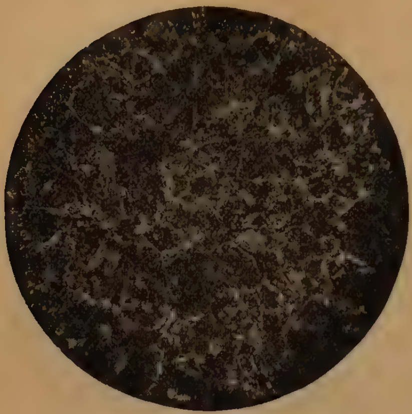
Fig. A.

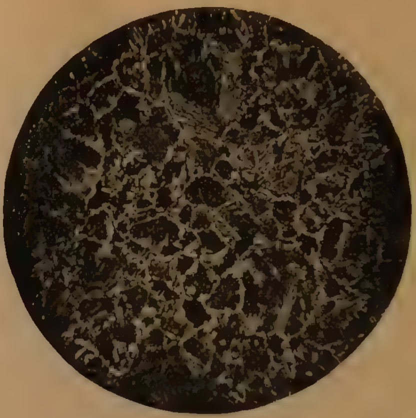
Fig. B.

Microstructure of test specimens at breech end. × 80.


Fig. C.

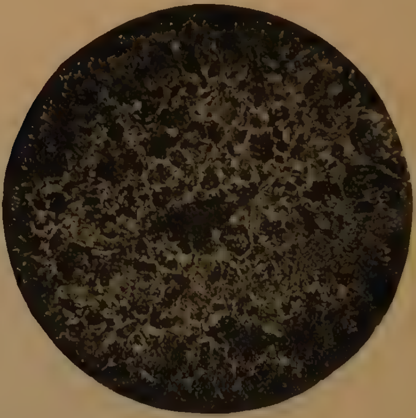
Fig. D.

Microstructure of test specimens at muzzle end.

Plate VII.—Gun Forging, Annealed, Quenched and Drawn. Carbon, 0.30 Per Cent.; Nickel, 3 Per Cent.

All Met Requirements: Tensile Strength, 90,000; Elastic Limit, 60,000; Elongation in 2 In., 18; Reduction in Area, 30. Fig. C gave Best Results, and Figs. A and C gave Highest Percentage Reduction of Area.

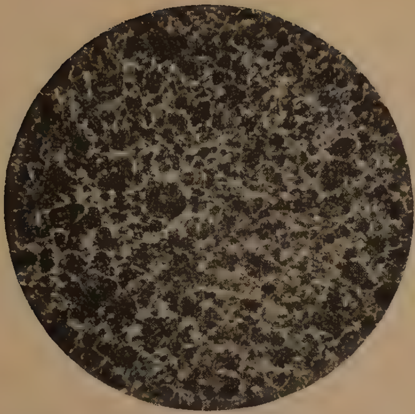

Fig. A.—Nickel steel. Carbon, 0.40 per cent. Tensile strength, 95,000; elastic limit, 62,000; elongation in 2 in., 24; reduction of area, 50. × 80.

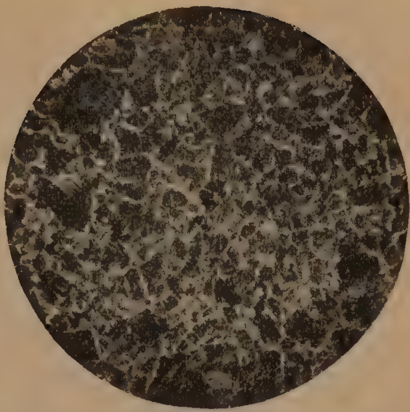

Fig. B.—Nickel steel. Carbon, 0.40 per cent. Tensile strength, 112,000; elastic limit, 70,000; elongation in 2 in., 21; reduction in area, 48. × 80.

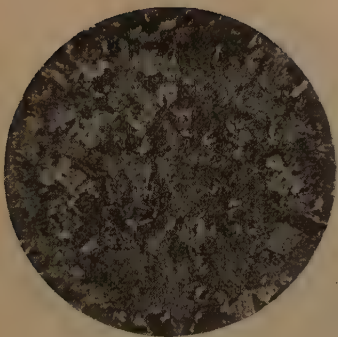

Fig. C.—Small nickel-chrome gun forging. Carbon, 0.40 per cent. Tensile strength, 104,000; elastic limit, 68,000; elongation in 2 in., 22.9; reduction in area, 58. × 82.

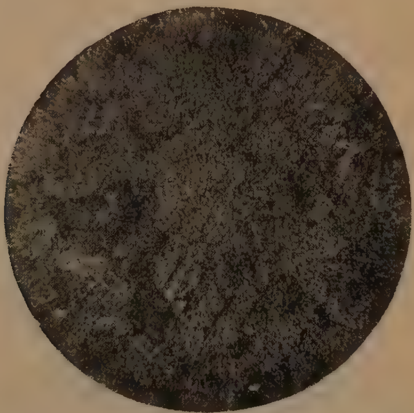

Fig. D.—Rare smaller forging. Tensile strength, 105,000; elastic limit, 63,000; elongation in 2 in., 25; reduction in area, 54. × 155.

Plate VIII.—Characteristic Microstructures Found in Nickel Steel for Guns—Annealed, Quenched and Drawn.

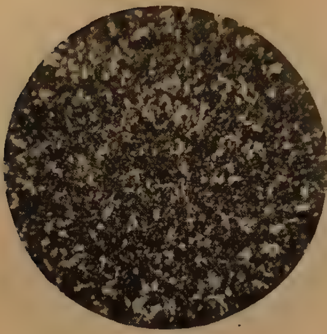

FIG. A.—Small forging of 3.5 per cent. nickel steel. 0.35 per cent. carbon. Tensile strength, 95,250; elastic limit, 60,000; elongation in 2 in., 24; reduction in area, 54. × 82.

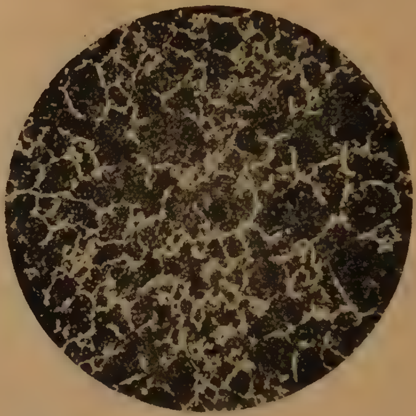

FIG. B.—Gun steel. 0.50 per cent. carbon. Tensile strength, 98,000; elastic limit, 58,000; elongation in 2 in., 18; reduction of area, 30. × 80.

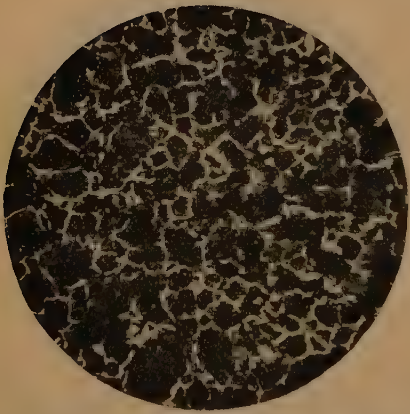

FIG. C.—Gun steel. 0.45 per cent. carbon. Tensile strength, 99,000; elastic limit, 57,000; elongation in 2 in., 25; reduction in area, 54. × 80.

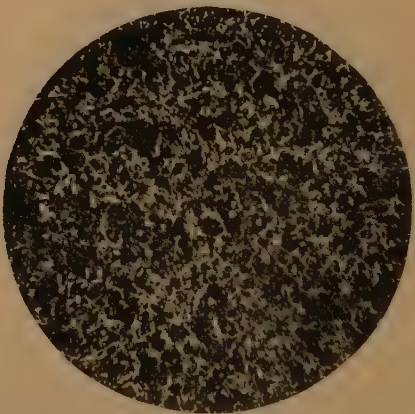

FIG. D.—Gun steel. 0.50 per cent. carbon. Tensile strength, 105,000; elastic limit, 63,000; elongation in 2 in., 26; reduction in area, 53. × 80.

PLATE IX.—CHARACTERISTIC MICROSTRUCTURES FOUND IN STEEL FOR GUNS—ANNEALED, QUENCHED AND DRAWN.

have exhibited excellent results. It is believed that if greater care were exhibited in the heat treatment of castings to insure heating them and holding them at just above the upper critical, taking into careful consideration the chemical composition, and cooling them at such a rate through the critical range as will give the most desirable properties, physical properties vastly superior to those at present obtained would be the result.

I will now give some examples of the microstructure found in various ordnance castings. The specimens examined are prepared by sawing off the test specimens transversely in the threads and polishing and etching by the usual methods.

Plate I shows the structure of a class "B" casting in the green, as cast, and before any heat treatment. Fig. *C* shows a complete ingot crystal at a low magnification (21). Fig. *D* shows the structure of a casting containing 3.5 per cent. nickel, in the green.

Plate II shows some photographs taken from physical test specimens from a Class "B" ordnance casting weighing about 500 lb. This casting was accepted on the test specimen, which showed a tensile strength of 73,500 lb., a yield point of 44,500, with 24 per cent. elongation in 2 in., and 38.7 per cent. reduction of area. It was a simple steel casting of about 0.24 carbon which had received an annealing treatment from a temperature supposed to be just above its critical range.

Figures *A* and *B* show the structure at 80 magnification. The totality of the ferrite appears to have been rejected, and the pearlite under high magnification shows the normal laminated structure.

Figures *C* and *D* show the same structure at a magnification of about 21, including a greater area in the field. It will be seen that the structure resembles very closely an ingot structure which has apparently not been broken up by the treatment. It is thought that the treatment temperature was probably less than reported, and was not high enough or sustained long enough to break up completely the original structure. It seems remarkable that such satisfactory physical results should have been obtained.

Plate III shows two specimens taken from 0.30 carbon, nickel-steel castings which contained between 3.25 and 3.5 per cent. of nickel and about 0.2 per cent. of chromium. These castings received a double annealing treatment and showed from 93,000 to 102,000 lb. tensile strength with a yield point of from 61,000 to 64,000 lb., with 22 per cent. elongation and 38 to 40 per cent. reduction of area. In the structure of these castings it appears that the alloys have caused a part of the ferrite to be retained in solution, as the dark areas are not pure pearlite, appearing to be in the transitory stage with some sorbite.

Plate IV shows a structure taken from another physical test specimen of practically the same composition, which gave a tensile strength of

91,000 lb., a yield point of 64,000 lb., elongation of 23 per cent., and reduction of area of 41 per cent. In this case there appears to be a greater amount of ferrite and its segregation appears in the nature of a ghost. The pearlitic structure is not characteristic, as the pearlite grains are so small.

Figure *A* shows the structure at a magnification of 80, and Figs. *B* and *C* at 32. The ghost lines are most prominent in the lower magnification.

Figure *D* shows the same specimen after it had been heat treated in the small furnace in my office. It was heated in one hour to a temperature of 1,500°F., held at this temperature for one-half hour and cooled in the furnace in two hours. The ghost-line structure seems to have disappeared, and the ferrite is more evenly distributed. It is not known, of course, what physical properties would be exhibited by such a structure.

In Plate V, Fig. A shows the structure exhibited by a test specimen of a class "B" casting, which passed physical test, giving the following results: tensile strength, 70,500 lb.; yield point, 36,000 lb.; elongation, 27.5 per cent.; and contraction, 49.2 per cent.

This was a 0.26 carbon, simple steel casting, and had been held at a temperature supposed to be 1,650°F. for 22 hours and cooled for 17 hours. It presents the characteristic structure of an ingot, which is noticeable by comparison with Plate I. It is not understood how such good results were obtained in elongation and reduction of area with such a poor microstructure.

Fig. B shows a similar casting which gave similar results. The upper critical temperature was probably never reached in the treatment of these castings.

Fig. C shows the structure of a 0.30 carbon, nickel-steel casting containing about 2 per cent. of nickel. The physical results shown were: tensile strength, 89,000 lb.; yield point, 54,500 lb.; elongation, 18 per cent.; and reduction of area, 29.2 per cent.

Long heating in a single treatment, and slow cooling, evidently caused the ferrite to collect in large grains.

From the very poor structure exhibited in the examination of numerous ordnance castings, I believe that great progress will be made in future in the development of more satisfactory and uniform castings through more scientific heat treatment. Probably at the present time the specification requirements are none too severe, so that with well-melted steel the fulfillment of the requirements is not a difficult matter.

Castings received and finally accepted for installation in the manufacture of ordnance, appear to be highly suitable for the purpose intended. It does not appear to be advisable to incorporate requirements which would compel the manufacturer to install expensive equipments

for more scientific treatment, thereby increasing the cost. But a special grade might be added to the specifications which would cover these points.

At present the greatest difficulty in the development of ordnance castings appears to be the preventing of blow holes, segregations and shrinkage defects.

As regards the microstructure noted in the examination of ordnance castings, it may be said in general that a pearlitic structure is exhibited; that when an alloy steel is used the totality of the free ferrite is not rejected and there is a predominance of the darker areas. The usual impurities are noted throughout the structure always in the form of rounded areas, the slate-colored impurities being silicate of manganese, whereas the manganese sulphide has the characteristic dove color.

Before taking up the more important question of forgings for ordnance purposes, it may be of interest to remark that while the Bureaus of Steam Engineering, and Construction and Repair have chemical laboratories, where analyses are made of drillings taken from castings and forgings by their inspectors, the results to be passed on before acceptance, the Bureau of Ordinance has not. A copy of all analyses taken by the manufacturers is always required, and this has been considered sufficient. Check samples have been taken from time to time, and in cases of doubt, and sent to the nearest government chemist for analyses. The results have not seemed to warrant making our own analyses of all material. This course is probably rendered doubly safe by the fact that nearly all ordnance material of importance is subjected to a ballistic test of some nature before final acceptance, and much latitude is therefore allowed the manufacturer in the chemical composition used.

The specifications for steel and alloy-steel gun forgings, except for a few very minor details, are the same for both the Army and the Navy. They are issued in the form of a pamphlet, revised from time to time, which goes into the requirements for manufacture and inspection in considerable detail.

The manufacturer is required to furnish reports of all analyses and physical tests made by him during the manufacture. He is required to furnish reports in advance of each operation in the manufacture in order that any operation desired may be witnessed. The government stamp is placed on each forging, transferred as necessary, and carried through to completion.

The term "gun steel" is defined as a simple carbon steel, and the term "alloy steel" as a simple carbon steel with such percentage of alloy added as is necessary to obtain the required physical qualities. Thus, if the required physical qualities could be obtained with a simple carbon steel, the manufacturer would be at liberty to do so, and the question has been considered from time to time of specifying a minimum allowable percentage of nickel in alloy steel, in order to impart certain desired

properties and to prevent a too rigid heat treatment to obtain the required properties. At present the only definite requirements as to composition are that the allowed percentage of sulphur and phosphorus shall not exceed 0.05. In examination of the microstructure of gun forgings, sulphur is usually seen in the form of manganese sulphide, which will be discussed later, and phosphorus segregations have been suspected in certain light-colored areas which have been brought out by the use of Rosenhain's reagent.

The specifications require that from open-mold ingots the discard from the top shall be at least 30 per cent. of the total weight of the ingot, and from the bottom at least 5 per cent. Possibly it would be better if the specifications did not prescribe a definite amount of discard but required the discard in each case to be sufficient to obviate any chance of piping or marked segregations. It is my opinion that 5 per cent. discard from the bottom of large ingots is an insufficient amount, but as this would be developed in the testing, the amount in excess of 5 per cent. should be left to the discretion of the manufacturer. From fluid-compressed ingots there is a slight reduction in the amount of discard required.

It being understood that the object of forging, in addition to its necessity for shaping a piece, is also to impart to it a mechanical grain refinement to put it in a condition whereby heat treatment will attain the desired end, a minimum amount of forging reduction is prescribed. The specifications say that ingots must be of such size that when pieces are forged solid the area of cross-section of the ingot shall be at least four times the maximum area of cross-section of the rough forging, and when bored ingots are used, the wall thickness of the ingot must be reduced by forging 50 per cent. Some forgings are of such a size that it would not be possible to make them from the largest ingot and give them the amount of reduction prescribed by the specifications. Other forgings are of different size at each end, and the shaping of them of necessity requires a different amount of forging reduction. In many such cases it is necessary in the process of manufacture that the ingot, after being punched or bored, be upset a certain amount, and in other cases forgings are drawn out and then enlarged on a bar. In all such cases a record is made of the total amount the metal has been worked. In other words, the upsetting operation is also considered as a forging reduction, though it is not, of course, in strict accordance with the wording of the specifications. It must be recognized that the amount of forging reduction alone will not insure a mechanical grain refinement, and that the temperature at which the forging is completed, and the subsequent treatment of the forging, play a most important part. If the forging is stopped at too high a temperature and the metal allowed to cool slowly, there will, of course, be a grain growth, and in large masses of metal this cannot be entirely prevented, as the metal cannot be worked entirely down to the

critical range; but this is taken care of in the requirements under "Heat Treatment." Physical tests and metallographic inspection of the ends of forgings which have had a different forging-reduction at each end, have not shown any marked difference attributable to the amount of reduction. Most long forgings will show the best physical results in specimens parallel to their axes, and short hoops enlarged on a bar transversely. But most forgings which are worked in more than one direction mechanically show little difference between transverse and radial or longitudinal bars.

The specifications for treatment require that the manufacturer keep a record and furnish copies of same to the inspector, showing the length of time for heating, the time held at that temperature and the time of cooling of every operation in thermal treatment. It is also required that he have an accurate means of ascertaining the temperature used. It is also required that, when practicable, forgings must be rough machined before treatment. This is necessary to insure maximum penetration and uniformity in the treatment. The precautions to prevent too rapid heating and the length of time during which the forging is held at annealing temperatures, is required to be satisfactory to the inspector.

There is a provision also in the specifications requiring all forgings to be annealed from the temperature above the Ac_1 point after forging and before tempering. This provision for the preliminary annealing covers the point referred to above, but it is believed that the Ac_3 critical point should be prescribed in place of the Ac_1 to insure better results.

It is required that all gun forgings of length be heated for quenching in vertical furnaces, which it is believed insure a more even and uniform heat, and the forgings are required to be immersed in the quenching medium in the direction of their axes. Longer pieces such as tubes and some hoops are often distorted a slight amount in treatment and it becomes necessary, before final machining, to straighten them. The specifications allow no straightening after test for acceptance as to physical properties. It is also required that the straightening shall be performed at a temperature between 800 and 900°F. and that after straightening they shall be reheated to the same temperature and annealed.

I have seen physical tests made of some shafting, and also some gun forgings, before and after straightening at this temperature, and though these tests showed no difference in physical properties, yet reheating is thought to be a wise precaution, since probably at this temperature the metal is more pliable. I believe tests have been conducted on metal slightly heated which have shown a falling off in physical properties at that temperature, which are apparently restored when the metal is again cooled.

When ample allowance of test metal on each end of a gun forging is allowed by the manufacturer, and when, after heat treatment, trial-test bars show the forging to be in proper condition, it is submitted to the

inspector for official test. The test specimen used is a standard cylindrical bar 0.505 in. in diameter and 2 in. between measuring points. The number of test specimens taken from a forging varies. From the largest forgings, four specimens are taken from each end and a smaller number for the smaller sizes. The number of test specimens taken from an end should be a function of the area of cross-section and not of the length. An exception to this would be in the case of a very short ring in which it might be considered not necessary to test each end. Thus, if three tests from each end are required from a hoop of a certain length, there would be no occasion for taking more than three tests from each end of another forging, the length of which is two or three times as great; that is, assuming that three tests from an end are sufficient to demonstrate its uniform quality. All bars from gun forgings subject to bursting stresses are taken circumferentially, or rather tangentially.

It is required that the minimum distance of the axes of test specimens from the end of a forging shall be 1.5 in. for long forgings and 1.25 in. for all other forgings. It has been found necessary to prescribe a distance, because tests taken close to the face give a slightly different property where the metal has been quenched, the rate of cooling probably being affected by the radiating surface of the end.

The physical properties prescribed (tensile strength and elastic limit in pounds per square inch; elongation on 2 in. and reduction of area in per cent.) vary with the type of forging. For alloy-steel forgings they vary from 90,000, 65,000, 18 and 30, for tubes, to 95,000, 55,000, 18 and 30 for certain hoops. For gun steel they vary from 86,000, 46,000, 18 and 30 to 93,000, 53,000, 18 and 30.

It is believed that it would be very desirable to increase the requirements for reduction of area. In my opinion, that reduction of area is one of the most desirable requirements inasmuch as it is an indication of the amount of distortion which will result before rupture after the tensile strength has been reached, and it also shows the material capable of sustaining a fiber stress greatly in excess of the tensile strength before actual rupture of the metal occurs.

As will be shown later, the mass of large gun forgings is such that it is practically impossible to eliminate segregations and defects such as ghost lines or streaks from the forging. If the steel were pure, there would not be much difficulty with the modern scientific treatment in meeting the requirements of the specifications as to physical properties. But while nearly all the test specimens from a forging may give results considerably in excess of 40 per cent. reduction, there may be one or more bars which will give less, caused by the aforesaid conditions, so that the increase in this requirement would probably be objected to strongly by the manufacturer, unless some provision were made to take care of bars which might fall short.

In figuring the elastic limit and tensile strength in tests, reference is always made to the original cross-sectional area which in the test bar used is 0.2 sq. in. Thus the stress per square inch is obtained by multiplying the actual load on the beam of the testing machine by 5. Even at the point when the tensile strength has been reached, there has been a slight reduction of area so that the actual sustained load on the test bar at that point has been greater than the recorded tensile strength. In the same manner if the actual sustained load on the beam could be measured after the test bar has necked down and just before rupture, it would be found to be very much greater, due to this reduced area. This sustained load at rupture, or rupture stress as it has been called, seems to me to be of extreme value, and it would be of value to develop some practical method of determining and recording it.

The elastic limit is required to be determined by the use of the micrometer extensometer. This requirement may be waived, provided the testing machine used is shown to have no difference between the elastic as determined by the drop of the beam and the micrometer readings for the grade of material in question. Of course this would only be true in machines where friction and inertia of moving parts are inconsequential.

Loads below the elastic are required to be applied in accordance with a table in the specifications, and for each load the actual extension must be recorded. It is required that the accuracy of the testing machine shall be verified at least once each year.

Three official submissions for physical tests are allowed on each forging, which may be either without or following retreatment. In case one bar from an end on physical test fails in extension and breaks within 0.5 in. of the measuring point, or in case one bar from an end breaks and the fracture shows that it failed because of inclusions of sand or slag, an extra bar is allowed immediately in wake of, and nearer the finished metal than the failing bar.

The tensile specimens are the only ones taken for physical tests from material for gun forgings, and are all that have been considered necessary, in view of the fact that the final test consists in the proof firing of the gun. In the test of material used in foreign gun construction, it is understood that the physical properties are generally lower than those prescribed for ours, but certain other tests are required in addition, such as mandrel tests, bending tests and fatigue tests. These last tests, when autographic vibratory machines reach perfection, may prove of great value and may demonstrate the unsuitability of material which would not be shown by other tests.

During the past year or so, for the purpose of collecting information, we have been examining and photographing the structures of many gun forgings. The test specimens are usually sawed off in the threads and the button thus formed is polished and etched in the usual manner. A

number of characteristic structures found in what are considered to be excellent forgings, are shown in Plates VIII and IX.

The use of nickel appears to insure a finer grain as well as to prevent the precipitation of the same amount of ferrite, that would result in plain gun steel.

Microscopic and macroscopic inspection has proved of the greatest value in investigating the character of defects in large forgings which are often noted in machining. These defects have been referred to in the specifications as follows:

> "In the case of streaked forgings, that is, forgings showing striations of different-colored metal, careful examination will be made to see if there is any lack of continuity of the metal along such streaks. Such lack of continuity determined by the breaking or parting of the chip in turning or in chiseling in the direction of the streak, or in any other manner, will cause the rejection of the forging.
>
> "Slag pockets and sand splits or cavities containing particles of slag, sand, or other foreign material will be treated as local physical defects in the steel, the seriousness of which will depend upon their location in the forging, their number per unit area of surface, their size and form, etc. If isolated and small their presence in a forging otherwise sound and satisfactory will not be deemed important; if in clusters or extending over a considerable surface, indicating a porous condition of the metal, the forging will be rejected."

Defects commonly known as streaks and sand splits exist to a certain extent in all large forgings, where the ingot from which they have been made has been one of considerable mass, but every effort is made toward reducing them to a minimum.

The defects known as sand splits are detected only in the inspection of surfaces smooth machined, and are found generally in the bore of large forgings after the last machining operation. They vary in size from the most minute specks to small pockets as long as ½ in. At times larger ones have been observed. These splits, which are generally elongated in the direction the metal has been most worked in forging, contain inclusions which often in the larger ones may be picked out with a small penknife, and resemble sand. Some of them contain a harder inclusion more difficult to remove. They are certainly related to the microscopic inclusions of sulphides and silicates which are generally segregated in ghost lines or streaks.

In Plate VI, Fig. A, is shown, at a magnification of 80, a microscopic inclusion of such a size that it appeared on the polished surface of the specimen to the eye without magnification, as a pin point. This inclusion appears to contain three substances. One, which is slate-colored (the darker in the photograph), is believed to be silicate of manganese. The matrix, which is dove-colored, appears to be manganese sulphide. There is a third constituent (white in the photograph), which is not understood. This specimen was from a piece of extremely impure steel; and the dark boundary of the inclusion which appears in the photograph

is a cavity caused by ripping out a portion of the inclusion in polishing. There are also shown numerous spots of sulphide.

Streaks appear to be of two general classes. They are noted as faint lines in the bore and on the surface of machined forgings, where the metal appears slightly brighter or darker in color. Polishing the surface and etching with iodine solution in alcohol will bring them out clearly as will also sulphur prints.

Sulphur prints, which are often made to investigate forgings where segregations of sulphides are suspected, are made by polishing carefully the surface to be examined, cleaning it off very carefully with alcohol to remove all oil, then placing upon it, face down, a piece of bromide paper which has been immersed in a solution of hydrochloric acid. After a few seconds the acid will attack any segregations of sulphides, discoloring the paper, and the photograph so formed is then fixed in the usual hypo solution. These sulphur prints bring out clearly the streaks when the inclusions which predominate in the streak are sulphides. Many streaks and inclusions are noted, however, in forgings when no clear sulphur prints of them can be obtained. These inclusions, which are generally slate-colored, are hence believed to be silicates. It should be noted that a sulphur print can be obtained from the polished surface of any piece of steel inasmuch as the sulphur in the steel, if almost entirely in the form of manganese sulphide evenly distributed, will cause an impression on the sulphur print of many specks which may appear to be more serious than the condition warrants. When these specks are examined after etching with the usual solutions, under the microscope, their character is clearly brought out.

In the first class of streaks which I have noted, there is a marked predominance of ferrite and an extremely fine grain. On each side of the streak the grains increase in size until the normal sorbitic structure is noted. Scattered along the streak are generally found many microscopic inclusions of sulphides and silicates often in lines. The metal in these streaks does not appear to differ from that on either side of it as to machining, and scleroscope readings do not show any difference in the hardness. When the inclusions are arranged in lines they, and any lines or cracks joining them, are carefully examined.

In the worst of these streaks, where the inclusions are in such number and of such size as to cause an actual break in the continuity of the metal, their seriousness would be indicated by chipping out the metal and the minute breaks would be shown by examining with a small magnifying glass. The general character of one of these streaks is clearly shown in Plate VI, Fig. B, in which a line of very small inclusions is also shown. This was a small streak which extended through a physical test specimen which had passed test; and the only noticeable effect was a falling off in

reduction of area as compared to other bars from the same forging which did not show the streaked metal.

In the second class of streaks which I have noted, the grains are much larger and there is not so much ferrite predominating as on either side of the streak. On some of these streaks a difference in the scleroscope reading is noted and the metal is shown to be harder. The inclusions noted are of the same nature in both classes of streaks. These streaks are more broken and irregular. Their characteristic structure is shown in Plate VI, Fig. C. The small test specimen which this structure represents, exceeded all the physical requirements.

The question of streaks and sand splits is being constantly studied and all such defects, even though very slight, are made a matter of record. The final test of the gun of which these forgings are made, of course, is the proof-firing.

The photographs of streaked metal are of streaks of the less serious nature. In the more serious streaks, which are rare, the metal will break off in machining and actual breaks in the continuity of the metal will be noted on macroscopic inspection.

In Plates VIII and IX are shown characteristic average structures found in gun forgings. These structures in general consist of sorbite and ferrite intimately mixed and are logical structures following the treatments which they have received. Some structures appear to be pure sorbite, and almost amorphous, but these are generally the smaller alloy-steel forgings such as breech blocks and short hoops for the smaller guns. Gun forgings to meet the properties required after the preliminary anneal specified are quenched and drawn.

In Plate VI, Fig. D, also in Figs. A to D, Plate VII, is exhibited a peculiar structure which is often noted in gun forgings. The structure consisting of a network of ferrite and sorbite, shows much of the ferrite in definite parallel lines within the cells, in some cases having an almost triangular formation. At a high magnification this structure resembles much the structure of an ingot at a low magnification. It is thought to be due to the structure formed in quenching, running from troostite to martensite according to the composition, which has persisted after drawing, allowing the ferrite on rejection to mass along the lines of this structure. The physical test specimen from which Fig. D, Plate VI, was taken, gave excellent results.

Before closing this paper I will refer briefly to one other class of forgings which has caused concern in drawing up requirements, namely, the air flask of the torpedo. This forging is about 8½ ft. long and 21 in. in diameter, and contains the air chamber for propelling the torpedo. It has to sustain a very high internal pressure and of necessity must be as light as possible. For the test of the forging, two tensile tests are taken from each end. We have obtained a number of forgings which have

passed specifications requiring 120,000 lb. elastic limit with 12 per cent. elongation in 2 in., and 35 per cent. reduction of area. Various specifications on other contracts have varied from this to 105,000 elastic limit with 15 per cent. elongation and 40 per cent. reduction. To meet such specifications an extremely pure steel is necessary. It has been noticeable in such tests as I have witnessed, that a small and inappreciable amount of slag may cause a test bar to break off short in this material, whereas the same amount of impurities in material in a softer condition would have an inappreciable effect on the results obtained by testing. This point is thought to be an important one, which should be considered by designers.

In conclusion, I might say as regards metallographic inspection, that at present in the larger inspection offices we have a Tassin microscopic outfit which can be applied to forgings in the shop if desired, or set up in our laboratory for examining smaller specimens there. This microscope has a photographic attachment and we have a dark room so that we may develop our own photographs as taken. It is our general practice in examining specimens for polishing to examine before etching, then to etch lightly and examine, and then to etch deeper and examine. In examining under the microscope we generally get as big a field, or low magnification, as possible at first and then examine with our standard magnification of about 100; and if interesting points are noted we then examine with higher magnifications.

We have a portable polishing apparatus which we can carry into the shops and polish surfaces desired.

We also have a scleroscope which we use generally for experimental purposes.

It will be seen that our aim at present in using the microscope in inspection is to collect all the data possible and to gain additional information regarding defects which may be noted in the usual inspection. It appears that the microscope is continually finding new uses to the manufacturer and more manufacturers are taking it up each day. It is believed that it has a big future, both to the manufacturer and later, probably, in inspection.

Discussion

Albert Sauveur, Cambridge, Mass.—I think we must all agree with Lieutenant-Commander Cook when he says that there is a great deal of room for improvement in the annealing of steel castings and that better results would be attained if the annealing were conducted in closer agreement with scientific requirements. Whether even a carefully annealed steel casting can ever be made the equal of a steel forging of good quality is still a debatable question. Nevertheless, we all realize that very greatly improved results can be attained by careful annealing.

Lieutenant-Commander Cook refers to some requirements calling for annealing steel castings above the Ac_1 point. He is undoubtedly right where he says that this should be the Ac_3 point. Take, for example, a steel casting containing, say, 0.35 per cent. of carbon, having a very coarse structure made up of coarsely crystalline ferrite. If this is heated just above the Ac_1 point the ferrite remains, of course, just as it was before, while the pearlite is converted into a solid solution, austenite. Upon cooling, all this austenite is again converted into pearlite, or sorbite, while the coarse structure of the ferrite remains unchanged and the annealing has accomplished very little. It is therefore surprising that the requirement of annealing above the Ac_1 point should appear in these specifications.

Harold Earle Cook, Philadelphia, Pa.—The specifications to which Professor Sauveur refers are specifications for gun forgings, which prescribe a preliminary annealing above Ac_1. As I said, the recommendation has been made that this should be changed to Ac_3 or Ac_{2-3}. Professor Sauveur's remarks, however, apply to the casting specifications also, which have a little different wording, *i.e.*, that all castings must be annealed at a temperature above 1,500° F. In another place in the paper I have referred to this and said that by heating to this temperature and holding some steels there, such as, for example, an alloy steel of medium or high carbon, we obtain just the result we do not want.

Albert Sauveur.—I am very glad that the author has explained the mistake I made. I thought he was referring to the annealing of steel castings. Nevertheless, in my opinion, the treatment is quite as inaccurate for steel forgings as it is for steel castings.

J. W. Richards, So. Bethlehem, Pa.—This paper discusses a number of exceedingly practical questions and will well repay careful study and attention, as for example, the importance of vibratory tests. I notice on p. 256 it is specified that at least 30 per cent. of the weight of the ingot must be cropped from the top, and at least 5 per cent. from the bottom. That probably applies to ingots cast with the larger end down, and the question of safely reducing the amount of crop by such improved methods as, for example, casting with the large end up, is seen to be a very vital question of economy in steel manufacturing, because the cropping of 35 per cent. of the ingot very much increases the cost of manufacture and leaves room for possible improvements such as those mentioned in the paper by Emil Gathmann.[1]

The question of occluded slag in the metal also brings up another very desirable improvement in steel. You can adjust the composition

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[1] The Commercial Production of Sound, Homogeneous Steel Ingots and Blooms, *Trans.*, This volume, pp. 341 to 348.

of the steel to the point desired; and you can deoxidize it as far as possible, but there will still remain the refinement of eliminating from it the suspended oxides and silicates. This is the last refinement in the manufacture of steel and its importance is just being realized. The proper furnace for doing the work is undoubtedly the electric furnace, and the time is probably coming when almost all carefully refined steel will be put through the electric furnace for this purpose.

W. E. Ruder, Schenectady, N. Y.—Why is it required to crop off the lower 5 per cent. of the ingot?

Harold Earle Cook (communication to the Secretary*).—Most ingots for large forgings are bottom poured and the first metal flowing into the mold probably collects considerable dirt in the runner and solidifies as a bottom skin. In spite of this requirement impurities are often noted near the bottom end of forgings which might even indicate that the present requirement is insufficient. The cropping requirements refer only to open-mold ingots. For ingots produced by improved methods, such as fluid compression, we require less discard both at top and at bottom, but for an ingot cast with the large end up, the requirements for discard would be just the same. The requirements for discard are for percentage by weight.

Richard Moldenke, Watchung, N. J.—I would like to remark that the two points just brought out can also be applied to the production of gray iron castings. The steel ingot requires cutting off at the top to remove unsound portions. So also the gray iron casting—if I may call it a parallel case. The sand mold has to be so arranged that a feeding-head is provided, and this will contain the spongy, unsound metal corresponding to the ingot top. In many castings, such as rolls and pipe, special provision is made to machine off the top portions, to get away from this defective metal; similarly the question of entrained material, such as slag, in the steel ingot. This case exists in the foundry also, and it is necessary to melt as hot as possible, and allow the metal to stand in the ladle before using, in order that it may clear itself of this entrained material which is skimmed off before pouring.

J. W. Richards.—These entrained foreign materials are what the foundry men call dirt, are they not?

Richard Moldenke.—Yes, sir. Dirt is the name given to this material.

Leonard Waldo, New York, N. Y.—I would like to ask Lieutenant-Commander Cook whether chrome-nickel steels have been found satisfactory in present use in ordnance, and whether any macroscopic and

* Received Mar. 22, 1916.

microscopic researches have been made to show the fineness of the grain and the prevention of the precipitation of ferrite.

Harold Earle Cook.—Some small gun forgings have been made of chrome-nickel steels, which are quite similar to ordinary forgings under ordinary methods of manufacture and treatment and have given excellent physical results. The naval gun factory now has in operation a small openhearth furnace which has been making a number of small forgings, and various other alloys have been made experimentally with excellent results, but there appears to be some objection to the use of such alloys for very large forgings, as for example, forgings of 14-in. guns which weigh as much as 17 to 18 tons after rough machining. In the microscopic examination of nickel-steel forgings it has always seemed to me that there is less ferrite out in the network boundaries than in a simple carbon steel of the same carbon content and similarly treated.

Samuel L. Hoyt,† Minneapolis, Minn. (communication to the Secretary*).—I heartily concur with Lieutenant-Commander Cook in so far as he has considered metallography as including subjects other than the microscopic examination of metals. The name *Metallography* was first attributed by Osmond, and rather appropriately, to the microscopic examination of metals. Not until the scientific study of metals became the primary work and interest of a large and important group of investigators did the term *Metallography* come to be defined as embracing not alone microscopic examination but various other fields of interest as well. The term *Metallic Microscopy* has been accepted as the more accurate and concise naming of the microscopic examination of metals, and at present *Metallography* is widely if not generally recognized as the science of the properties of metals. It must be borne in mind, however, that the constitution and physical state (internal structure of metals and alloys) are regarded by the metallographer as being of prime importance as a foundation on which can be established the rational treatment of the physical and mechanical properties. So to say that "The internal structure of steel is dependent on its physical and chemical properties" seems to be stating things in the reverse order.

Certainly the author is on safe ground when he says that "It is believed that some day information gained by the use of the microscope may prove of great value." Our government is to be commended for having begun to supply some of its inspection officers with metallographic microscopes in order that at least a beginning may be made. It might be remarked that had this course been adopted a decade or so ago, as was done abroad, our government would be in a much more advantageous position at present, as a large and important customer, to bring about the

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* Received Feb. 21, 1916.

improvements in the steel-making industry of this country which are so needed.

The microscopic examination of the ordnance material as described by the author is very interesting and instructive, but I should like to confine my remarks to his discussion of the physical properties. He seems impressed by the fact that the microstructures as exhibited in Plate II and in Plate V, Fig. A, should have such satisfactory physical properties *as obtained from the tensile test.* First of all must be borne in mind what information the tensile test is able to reveal: The tensile test gives values of the static strength, the static resiliency (or better, the work of deformation up to rupture), and the static ductility. There is no particular reason why the structure shown in the above-mentioned microphotographs should not be both strong and ductile. This does not mean that a structure of the same nature, only somewhat worse, would not show a decrease in both the strength and ductility, but merely that the treatment has not been sufficiently harmful to be noticed in this respect. But it can hardly be assumed that these structures would stand up as well in service as would a more uniform and finer-grained structure, other factors being the same. The point is that these structures are deficient in toughness and that the static tensile test is not capable of giving an adequate measure of this property. The elongation and reduction of area not only are not at all times proportional to the toughness, but many times are equally as high in comparatively brittle material as they are in comparatively tough material.

It is now over 20 years since tests were first devised, principally by the French experimenters, which would permit a seriation of metals according to this particular property, toughness. These tests were generally made on notched bars and were used by the French Admiralty to test their armor plate as early as 1893. These experiments culminated in the reports of Barba, Frémont, Ast, and Charpy, to the International Association for Testing Materials, in 1901, 1906, and 1909, and that of Ehrensberger, Director of the Krupp Works, to the German Society for Testing Materials, out of which developed what is now known as the "notched-bar impact test."

For the purpose of illustrating the point concerning the physical properties corresponding to the above-mentioned microstructure, I wish to include a few data taken from the report of Ehrensberger:

The microstructure of No. 1, which was forged too hot, and No. 161, which was cast steel, both of which are strong and ductile, can be said (with sufficient accuracy for the present discussion) to correspond to the microstructure above mentioned. Judging from the static properties, there would be little choice between them and No. 2 which was properly forged and which would show a correspondingly finer grain under the microscope. The notch toughness as measured by the Charpy testing

Carbon Steels (Forged)

No.	T. S.	El. Lt.	Elong. $l = 11.3\sqrt{f}$	R. A.	N. T.
1	61,500	32,700	26.5	64	4.6
2	64,100	36,100	26.0	70	20.4
		Nickel Steel			
51	72,800	58,200	23.3	70	42.1
		Cast Steel			
161	68,000	37,700	22.9	51	3.7

T. S. = Tensile Strength in pounds per square inch.
El. Lt. = Elastic Limit.
Elong. = Elongation for a gage length equal to ten times the diameter.
R. A. = Reduction of Area.
N. T. = Notch Toughness in Mkg./sq. cm.

machine brings out the true superiority of No. 2 which was missed by the tensile test. No. 51 is a special steel and illustrates the fact that only on measurements of the toughness can an adequate comparison of the special steels and the ordinary steels be based. These facts have been recognized and incorporated into the tests of materials, such as are mentioned in the paper, for over a decade in Europe. It would be well for our government to do likewise. In this way, by selecting materials and heat treatments which would produce the desirable combination of static properties and toughness, there can be but little doubt that the construction materials used by the navy could be materially improved and brought to a more uniform quality.

The ballistic test, of course, can never be supplanted by laboratory tests, but for the purpose of developing materials and heat treatments it is but a cumbersome and expensive method. The microscope would undoubtedly prove of great assistance in this work, as is mentioned by the author.

It is difficult to understand what is meant by the remark that the difference between the true elastic limit and the yield point as determined by the drop of the beam depends, not on the material, but on the testing machine itself. It is true that the yield point is a function of the speed of testing but it is also a property of the material and in very carefully performed experiments it has been shown to differ from the true elastic limit by thousands of pounds per square inch.

Harold Earle Cook (communication to the Secretary*).—I agree with Mr. Hoyt as to his definition of metallography, and in his comments on that part of the paper. The microstructure of the steel is dependent on the chemical composition, method of manufacture, and heat treatment,

* Received Mar. 22, 1916.

and the microstructure is a measure of the success of these efforts to obtain the physical properties desired.

The remarks upon the relation of tensile tests and microstructure are of vital interest. For experimental tests the impact test, torsion test, and vibratory test are very valuable and I have believed that such forms of test would demonstrate the relative value of materials having the same tensile properties but different microstructures. But it has not been thought that testing machines for such purposes have yet reached a stage of development or standardization, or that sufficient data have been accumulated, to admit of the insertion in specifications of definite requirements for such tests.

I have considered the elastic limit, as the limit of proportionality, as determined by the micrometer extensometer or strain gage, and the yield point as determined by the drop or lift of beam. The former is the point where the first permanent set is detected by plotting a curve from extensions measured by the most sensitive instruments in use. This might be still lower if more sensitive instruments were devised. The yield point is the point at which the first appreciable stretch occurs and the amount it differs from the true elastic limit varies with the grade of steel and the accuracy of the testing machine. As to the statement in the paper, I had in mind the Emery testing machine in which in our grades of steel there is no apparent difference between the elastic limit as determined by the micrometer extensometer and the lift of the beam. We require the elastic limit for gun forgings and the yield point for unimportant forgings and castings.

G. F. Comstock,† Niagara Falls, N. Y. (communication to the Secretary*).—While this paper is on the whole of much interest and value, there are certain points mentioned which seem worthy of further explanation. The "broad ferrite bands," which the author has "noted from time to time extending across the field of the microscope" even after heat-treatment of the sample, have been seen by many other observers in castings, and their cause has been shown by Dr. Howe, Dr. Stead, and perhaps others, to lie in uneven distribution of phosphorus which does not diffuse thoroughly even after long annealing. The high-phosphorus areas or streaks always contain some sulphide inclusions, and Dr. Howe has shown by theoretical reasoning that it is the phosphorus which causes the ferrite to be precipitated in greater amount around these inclusions as the steel cools from the annealing temperature. This of course explains the reappearance of the broad ferrite bands after annealing.

The author's belief that "the primary ingot crystallization can be completely destroyed only by means of mechanical working," and not by

† In Physical Testing Laboratory, Titanium Alloy Mfg. Co.

* Received Mar. 9, 1916.

heat-treatment alone, is shared by me in a general way, but I believe, however, that the reason for this is not any inefficiency of heat-treatment in completely refining the ferrite-pearlite structure of the steel, but because heat-treatment has such a very slow effect on the phosphorus distribution, and no effect at all on the arrangement of the sulphides and other non-metallic inclusions. The original structure in regard to phosphorus and sulphides is broken up by mechanical treatment but not by heat-treatment, and this is the reason, in my opinion, for the essential superiority of forgings over heat-treated castings.

The specification requiring the enormous discard of 30 per cent. from the tops of ingots seems wasteful and somewhat antiquated, even when the exception in favor of fluid-compressed ingots is noted. A 30 per cent. discard is undoubtedly none too much to insure good results from ingots carelessly made by ordinary methods, but it has been proved repeatedly that a much smaller discard is ample for well-deoxidized steel carefully cast, especially if some anti-piping device such as Kenney's method or Gathmann's mold is used. The thorough deoxidation prevents segregation and blowholes, and it is then only necessary to control the pipe.

Lieutenant-Commander Cook's belief in the great importance of the reduction of area in a tensile test is interesting but it might be inquired why this property is considered more valuable than the elongation? I know from experience that when testing castings the reduction of area is very often a rather doubtful figure for the fracture may be irregular in area, or may have a lip extending outward on one side so that the micrometer cannot be applied at the point of smallest cross-section. Furthermore, practice may differ in two testing laboratories, as to whether the smallest diameter of the fracture-area should be taken to compute the "reduced area," or whether an average should be taken between the smallest and largest diameter of the fracture. On the other hand, the elongation is always a most simple property to determine with accuracy, and for that reason it would seem better to rely on it as the most practical measure of ductility, rather than on the more doubtful reduction of area.

The reduction of area is considered important because "it is an indication of the amount of distortion which will result before rupture after the tensile strength has been reached." The question arises here, why is it important that this distortion should be great? Of what use would a gun, structural bar, or machine part be after it had been distorted in service? Would not its useful life be ended the moment any appreciable distortion had taken place, even if it had not been stressed to the point of rupture? It surely must be admitted that permanent distortion is equivalent to failure in such material, and for this reason the amount of distortion taking place after the tensile strength has been reached seems to be of purely academic interest. I am not claiming that ductility in general is of no practical importance in this material, for it is unquestion-

ably necessary that the ductility be great enough to insure good shock-resisting properties, but this ductility can in general be shown most accurately by the elongation.

I cannot see any great difficulty in determining the "sustained load at rupture, or rupture stress as it has been called." This merely involves keeping the beam of the testing machine balanced up to the point of rupture, instead of letting it drop after the maximum load has been reached, and this can be done either with an automatic attachment on the machine or by hand if the test is run slowly enough. The area to be used in computing the unit stress in this case is of course the same as the reduced area, which is always measured after rupture. It might take two observers to record both the maximum load and load at rupture for the same test, but the need for the development of "some practical method" for this determination is hard to understand. If the determination is really "of extreme value," one wonders why it is not made more often. Personally, I cannot see the value of it, because, if the maximum stress is reached in practical use of a material, this stress will produce failure just as the failure is produced in testing, and the fact that distortion takes place does not mean that the material, in practice, will stand without failure a higher stress than the tensile strength determined in the usual way with a testing machine.

The author's remarks on the subject of sulphur prints might, in my opinion, be slightly misleading to anyone who would be guided by them in trying this method of examination for the first time. In this laboratory many hundred sulphur prints have been made, and the method used has been found not only very easy, but absolutely trustworthy. In no case has a sulphur print made here ever shown "many specks which appeared to be more serious than the condition warranted." Of course it is well known that all steel contains sulphur, therefore a sulphur print can be made from any piece of steel without exception, and every properly made print will show specks on it made by the sulphide inclusions in the steel. The mere presence of specks on the sulphur print does not therefore show anything as to the quality of the steel, but it is their arrangement that indicates this quality. If the print is well made, which is a very simple matter, there can be no question of the truth of its testimony as to the arrangement of the sulphides, and no chance of its appearance being "more serious than the condition warrants." Specks evenly distributed indicate homogeneous steel, and specks segregated in places into large heavy spots indicate segregated steel. That is the whole story. No other method of macroscopic investigation is so free from chances of misleading indications as the sulphur print method. But of course good practice is necessary here as with other things, and perhaps it may be of interest to outline the procedure found by experience in this laboratory to give the best results.

In the first place, careful polishing, in the sense that specimens are polished for microscopic work, is entirely unnecessary. The surface need not even be perfectly flat, and coarse emery cloth (No. 2 commercial) will do for the final polishing. Care must be taken, however, to keep the surface clean after polishing. There is no reason why this surface need be touched between the rubbing with emery cloth and the application of the bromide paper (Azo paper is cheaper and gives a print of better contrast), except to brush off the dust. No cleaning with alcohol is ever found necessary here. The steel must be allowed to cool after polishing, however, or the paper will slip on it, giving a blurred print. "A few seconds" is entirely too short a time of contact to make a good print, for it takes several seconds to press down the paper smoothly all over the steel surface to ensure good contact everywhere, and in order to get a good, even print the bulk of the printing action must take place after the actual applying of the paper is entirely completed. In this laboratory 3 per cent. sulphuric acid is used, and the first print on a polished section is allowed about 1 min. of contact. This print is usually not very good, as the time of contact to secure proper intensity is not enough compared to the time necessary for actually applying and smoothing down the paper. A second print is then made after washing and drying the steel, and judging from the darkness of the first one, a period of contact of from 2 to 5 min. is allowed. A third print is allowed from 4 to 10 or 12 min., and the second and third prints are usually sharply defined and should be identical, thus checking each other in regard to possible air bubbles which cause white spots, or hypo stains, rust spots, etc., which are dark brown or black. With reasonable care and a little experience these troubles may easily be absolutely avoided.

One more point that should be mentioned is the author's identification of all "dove-colored" particles in steel as manganese sulphide. He is evidently not familiar with the fact that iron oxide has practically the same light-gray color in a polished section, and it is no easy matter to distinguish between the two substances. While the sulphide inclusions are not always of the same color, they are as a rule a trifle lighter than oxide, and when steel has been worked they are thinner and more pointed in shape. I have found that with his method of polishing, oxide inclusions stand out more in relief from the metal, and are much more liable than the sulphides to contain pits, or in other words they do not polish so well. Some rather complicated tests have been suggested to distinguish definitely between these two kinds of inclusions, but I have not found any that are of much practical value with the microscope. Of course it makes little difference whether a small isolated inclusion is a sulphide or an oxide. In the case of a large group or streak of inclusions, whose effect may be harmful, a sulphur print will show very easily which they are, for sulphides will cause a large black spot on such a print, while

oxides will leave a mark or streak that is white, or at least lighter than the rest of the print. Also, oxides are always surrounded by decarbonized metal, which can readily be recognized by microscopic examination with low magnification after etching. Although the reproduction Fig. A, Plate VI, of the author's photomicrograph of a large inclusion is not clear enough to permit any accurate judgment to be formed as to its composition, the suggestion is offered that the light-gray portion and the surrounding small spots may be oxide, and the third (white) constituent may be metal formed by the reduction of this oxide by carbon from the surrounding steel. I have noticed such action in cases of castings which tore apart while hot in the molds, admitting oxide into rather deep cracks in the hot metal.

Vacuum-Fused Iron with Special Reference to Effect of Silicon

BY T. D. YENSEN,* B. S., M. S., URBANA, ILL.

(New York Meeting, February, 1916)

I. Introduction

It is safe to say that of all the different materials that go to make up electrical machinery, iron is the most important. Upon its magnetic and electrical quality depends not only the efficiency of the machine but, to a very large extent, the cost. Aside from the mechanical strength, there are three characteristics of iron that are of importance in this connection, *viz.*, magnetic hysteresis, electrical resistance, and magnetic permeability. In the homopolar machine, where the magnetic flux is unidirectional, permeability is the only important factor, but in all other machines the flux is reversed in one or more parts and causes losses due to hysteresis and eddy currents. The eddy-current loss depends largely upon the electrical resistance of the iron; the higher the resistance the lower the loss, under otherwise identical conditions. These hysteresis and eddy-current losses cause heating of the machine, and as they depend upon the magnetic density in the iron, this density must be regulated so that the heating shall not be excessive. If the hysteresis and eddy-current losses and the permeability of the magnetic circuit will permit an increase in the magnetic density, or, in other words, a decrease in the amount of iron, this in turn will mean a decrease in the amount of copper needed, with a consequent decrease in the copper loss. It is thus seen that, indirectly as well as directly, the above characteristics have an important bearing on both the efficiency and the cost of electrical machinery. In general, it may be said that iron for electrical machinery should have the following characteristics: (1) low hysteresis loss, (2) high electrical resistance, and (3) high permeability. These are the characteristics that investigators have been striving to attain since electrical machinery first came into use.

Until the beginning of the present century, Swedish charcoal iron was regarded as the best grade of iron for magnetic purposes, and even now it ranks high among the various grades used. About that time Sir Robert Hadfield[1] produced a number of iron alloys that revolution-

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[1] Barrett, Brown and Hadfield: *Scientific Proceedings of the Royal Dublin Society*, vol. vii, Sec. 2, Part 4, p. 67 (January, 1900). *Journal of the Institution of Electrical Engineers*, vol. xxxi, p. 674 (1901–02).

ized the iron industry as far as magnetic iron was concerned. His 2.5 per cent. silicon alloy and 2.25 per cent. aluminum alloy showed a higher permeability and gave a lower hysteresis loss than the purest Swedish iron. Perhaps of even more importance was the high electrical resistance of these alloys, reducing the eddy-current loss to one-third, or less, of that of the Swedish iron under identical conditions.

Since the time of the appearance of Hadfield's alloys comparatively little improvement has been made in the magnetic quality of iron and iron alloys. What has been done has been mostly in the way of modifications of Hadfield's silicon steel. In 1906 Prof. C. F. Burgess, of Wisconsin, commenced a series of investigations upon the magnetic and allied properties of electrolytic iron and its alloys with other elements. While the improvements made by Burgess were not remarkable, his investigations, I think, pointed out the direction in which improvements might be expected. Burgess and his associates melted their pure iron in a carbon resistance furnace in an atmosphere of carbon monoxide under ordinary pressure. It was found that by this method of melting the iron absorbed about 0.1 per cent. carbon, evidently by decomposition of the CO gases. According to Barrett[2] this carbon content should raise the specific electrical resistance of the iron from 10.1 to 10.6 microhms. However, the resistance for Burgess' purest iron was found to be 12.5 microhms, showing that the iron had absorbed some other impurity besides carbon. There is little doubt but that this other impurity is oxygen, because if carbon is absorbed by the decomposition of CO, the oxygen liberated would naturally combine with the iron.

The experiments on pure iron recorded by Burgess were used as the starting point of the investigation commenced by the author four years ago. Doubly refined electrolytic iron, deposited according to the methods used by Burgess, has formed the basis of the investigation. It was deposited from Swedish charcoal iron. The raw material and the product had the following composition:

	Swedish Iron, Per Cent.		Doubly Refined Iron, Per Cent.
C	0.1630	C	0.006 to 0.01
Si	0.0320	Si	0.01
S	0.0002	S	trace
P	none	Fe (by difference)	99.98
Mn	none		
Fe (by difference)	99.8000		

The iron was placed in crucibles made from electrically fused magnesia, and these crucibles were in turn placed in graphite crucibles and securely covered. At first a Hoskins resistance furnace was used for melting of the iron, but this was abandoned after a large number of attempts had

[2] *Proceedings of the Royal Society of London*, vol. lxix, p. 480 (1902).

been made to keep the iron from being contaminated. If the crucibles were left exposed in the furnace the iron was so badly oxidized that it cracked to pieces under the hammer in the attempt to forge it. If buried in crushed carbon the iron absorbed from 0.05 to 0.15 per cent. carbon, confirming the results of Burgess. A photomicrograph of the iron thus produced is shown in Fig. 1. In order to overcome the difficulties above referred to, a vacuum furnace was constructed, modeled after the Arsem type of furnace. The inside parts consist entirely of graphite, cut from solid graphite electrodes, with the exception of the water-cooled copper tubes that serve as supports as well as leads. A Geryck pump is capable of maintaining a pressure of 0.5 mm. of mercury or less with 500 or 600 grams of molten iron in the furnace. The state of the charge may be observed through a mica window in the top.

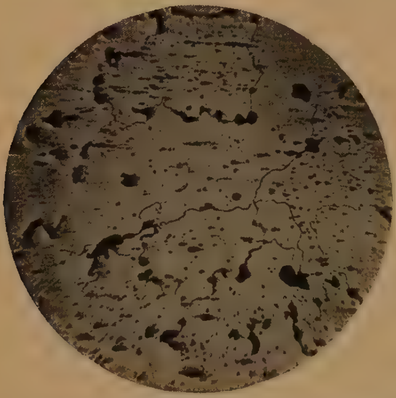

Fig. 1.—Electrolytic Iron Melted in an Atmosphere of CO Gas under Atmospheric Pressure (Compare with Fig. 11). Magnification 40 Diameters. Etched with Picric Acid.

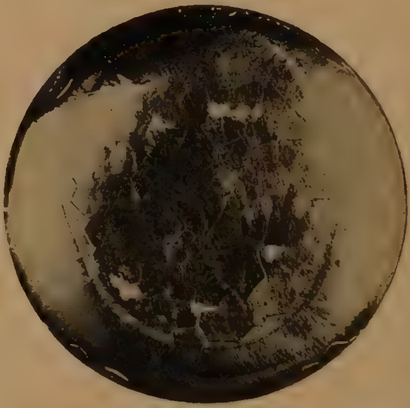

Fig. 2.—Top Surface of an Ingot of Pure Iron, Melted in Vacuo.

This method of melting took care of the contamination (see Fig. 11), and the iron, after being permitted to cool in the furnace, was perfectly bright when removed (Fig. 2). In magnetic quality, too, it rose far above anything known at that time.[3] The maximum permeability for this pure vacuum iron was found to be 19,000 as compared with 6,000 to 8,000 for the best commercial iron used at the present time. The hysteresis loss, too, was found to be much lower than for ordinary iron. An iron with very promising characteristics was thus produced. If the electrical resistance of this iron, which in the pure state is very low, could be increased without impairing its magnetic properties, an ideal iron for electromagnetic purposes would be produced.

In experimenting further with the vacuum iron, alloying it with small

[3] See *Bulletin No. 72, Engineering Experiment Station, Illinois University* (1914).

quantities of carbon[4] in one case, and boron[5] in another, the fact was revealed that, in spite of very careful chemical cleaning of the electrolytic iron previous to melting, the iron contained about 0.4 per cent. of oxygen in the form of some oxide of iron. This was shown by the disappearance of the equivalent amount of these alloying elements. In the case of carbon, this element would not combine with the iron unless added in quantities above 0.3 per cent. Assuming that the carbon was oxidized to CO, this amount of carbon corresponds to 0.4 per cent. of oxygen. Thus carbon would appear to be the ideal reducing agent for iron oxide *in vacuo*. The danger is, of course, that too much carbon may be added, leaving a small amount to combine with the iron, with disastrous effect upon the magnetic quality. Boron differs from carbon in its behavior upon iron, in that only part of it is oxidized by the iron oxide, while the remainder combines with the iron. The effect of the combined boron upon the magnetic properties of the iron—and upon some of the mechanical properties as well—is quite similar to that of carbon. Both elements should, therefore, be thoroughly eliminated from iron intended for high-permeability, low-hysteresis purposes. The results of these experiments suggested that if an alloying element could be found that had a strong reducing power at the same time as it had no lowering effects upon the magnetic quality, the magnetic properties of the pure vacuum iron might be improved even above what had already been attained. Silicon suggested itself as the most promising element for this purpose. The results given in this paper will show that these expectations have been fully realized. Certain percentages of silicon added to pure iron not only raise the electrical resistance to a desirable degree, but also raise the magnetic properties to a point that was not thought possible a year ago. While it is the magnetic properties that make these iron-silicon alloys particularly important, their mechanical properties, too, present characteristics that are of great interest, and these will be discussed in detail; a number of photomicrographs are also reproduced.

II. Material, Apparatus and Methods[6]

1. *General*

As already stated, the iron used as the basis of the investigation consisted of electrolytically refined iron, containing less than 0.01 per cent. carbon and about 0.01 per cent. silicon. Before being used, the iron was crushed, cleaned with HCl, distilled water and alcohol, and then dried

[4] *Bulletin No.* 72, *Engineering Experiment Station, Illinois University*, pp. 40 to 42 (1914).

[5] *Bulletin No.* 77, *Engineering Experiment Station, Illinois University* (1915).

[6] Details with regard to material, apparatus and methods used are given in *Bulletins No.* 72 *and No.* 83, *Engineering Experiment Station, Illinois University.*

by means of ether *in vacuo*. About 600 grams of this cleaned iron was then placed in a fused magnesia crucible together with the desired portion of the alloying element, covered with a magnesia cover and placed in the vacuum furnace where it was melted under a finishing pressure of 0.5 mm. of mercury. The ingots, after being allowed to cool in the furnace, were heated in an ordinary coke forge and forged into rods about ½ in. (1.25 cm.) by 20 in. (50 cm.) under a steam hammer. No contamination took place in this operation. From these rods the test pieces were prepared; one rod 14 in. (35.5 cm.) long for the magnetic and electrical tests; two test pieces, 2½ in. (6.3 cm.) long, with threaded ends, for mechanical tests; and one small specimen for the metallographic investigation. Magnetic and electrical tests were made and photomicrographs were taken after the following heat treatments:

(*a*) As forged.
(*b*) Annealed at 900°C. Cooled at a rate of 30° per hour.
(*c*) Annealed at 1,100°C. Cooled at a rate of 30° per hour.

Mechanical tests were made as forged and after annealing at 970°C. A few specimens were annealed at 1,100°C. The annealing was done *in vacuo* so as to preclude any possible contamination by gases, the vacuum part of the furnace used for this purpose consisting of an "electroquartz" tube with mercury-sealed ends.

2. *The Magnetic Testing*

The Burrows compensated double bar and yoke method was used for the magnetic testing, as this is the method now generally adopted whenever accurate testing is desired. Briefly stated, the apparatus consists of one main coil, and one auxiliary coil, separately controlled, and four compensating coils, connected in series. By means of three secondary coils the magnetic flux can be investigated at different points of the magnetic circuit and equalized by adjusting the currents in the magnetizing coils. With no leakage of flux, the magnetizing force at the middle of the main coil is

$$H_T = 0.4\,\pi\,N_T I_T$$

where N_T = number of turns per centimeter of main coil

I_T = current in main coil in amperes.

An error is introduced here on account of the effects of the ends of the various coils on the magnetizing force at the center of the main coil. With the silicon rods of highest permeability the maximum compensating current was 30 times the current in the main coil, causing a theoretical error in H_T of + 2.3 per cent.[7] However, experimental evidence seems

[7] See *Bulletin of the U. S. Bureau of Standards*, vol. vi, No. I, Reprint No. 117.

to show that the actual errors are larger than the theoretical considerations would lead to, and may in the extreme cases amount to eight times the theoretical error.[8]

Another source of error in the magnetic testing is mechanical strain caused by clamping the rods in the yokes, but this can be avoided by careful clamping.[8]

The apparatus was calibrated from time to time by means of an air coil, and also by means of rods submitted to the Bureau of Standards for standardization. The results obtained at the University check very well with those obtained by the Bureau and also with results obtained by the calibration laboratories of two of the large manufacturing companies.

3. *Photomicrographs*

These were obtained by polishing in the usual way, using jewelers' rouge for the final polishing. Great care had to be exercised with the higher silicon alloys. Numerous small spots would appear on the surface of some of the specimens, too numerous to be caused by graphite, as the carbon content amounted to only 0.01 to 0.02 per cent. By careful polishing the spots were in most cases removed, as seen by the photomicrographs.

The etching was generally done by means of saturated picric acid in alcohol. In a few cases, however, a 10 per cent. solution of nitric acid in alcohol was found to give more satisfactory results.

The highest silicon alloys, containing 4.92 and 6.57 per cent., respectively, had to be washed with dilute hydrofluoric acid to remove the thin film of a silicon composition that invariably formed on the surface after etching.[9]

4. *Mechanical Testing*

The mechanical testing consisted in obtaining the yield point, ultimate strength, the elongation before the specimen had commenced to "neck," the ultimate elongation, and the reduction of area at the point of fracture. The testing was done on an Olsen 10,000-lb. testing machine, in which the test pieces were secured between screw sockets. The load was applied uniformly by means of an electric motor, and in most cases the yield point was detected with certainty. The test pieces from the 6.57 per cent. alloy could not be threaded on account of their brittleness and had to be tested between flat grips.

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[8] See *Bulletin No.* 83, *Engineering Experiment Station, Illinois University*, 1915.

[9] L. Guillet: *Revue de Métallurgie*, p. 46, 1904.

III. RESULTS

1. *Chemical Properties*

Table I gives a complete list of all the alloys made, together with the chemical analysis for silicon. From the amount of silicon added, the percentage lost in terms of the weight of the ingot has been calculated and listed in the fourth column. The only elements that are present

TABLE I.—*List of Alloys*

Specimen No.	Silicon Added, Per Cent.	Silicon Content as per Chem. Anal., Per Cent.	Silicon Lost, Per Cent.	Remarks
3–51	0.000	About 0.001	....	These rods were discarded after annealing at 1,100°C. on account of oxidation due to a leak in the furnace.
3–52	0.000	About 0.001	....	
3–53	0.000	About 0.001	....	
3–54	0.000	About 0.001	....	
3–55	0.000	About 0.001	....	
3 Si 05	0.092	0.068	0.024	
3 Si 06	0.185	0.148	0.037	
3 Si 07	0.276	0.242	0.034	The magnetic data for these rods, after annealing at 900°C. were taken before the effect of strain due to tight clamping of the yokes was noticed. It was impossible to retest these rods annealed at 900° because they were already annealed at 1,100°. Data for the 1,100° annealing were taken without strain.
3 Si 08	0.368	0.309	0.059	
3 Si 09	0.460	0.400	0.060	
3 Si 10	0.551	0.472	0.079	
3 Si 11	0.643	0.563	0.080	
3 Si 12	0.734	0.673	0.061	
3 Si 13	0.825	0.698	0.127	
3 Si 14	0.916	0.822	0.094	
3 Si 15	0.138	0.064	0.074	
3 Si 16	0.046	0.010	0.036	
3 Si 17	0.230	0.230	0.000	
3 Si 18	1.810	1.741	0.069	
3 Si 19	2.690	2.550	0.140	Not forgeable. Crushed into mass of crystals.
3 Si 20	3.560	3.580	−0.020	Flaw in center of rod after forging.
3 Si 21	0.092	0.048	0.044	
3 Si 22	0.185	0.091	0.094	
3 Si 23	0.276	0.205	0.011	
3 Si 24	2.690	2.570	0.120	Not forgeable. Same as 3 Si 19.
3 Si 25	3.560	3.400	0.160	
3 Si 26	2.260	2.180	0.080	Contaminated in melting.
3 Si 27	3.125	2.730	0.395	
3 Si 28	4.410	4.440	−0.030	
3 Si 29	5.230	4.920	0.310	
3 Si 30	8.420	8.550	−0.130	Not forgeable.
3 Si 31	2.260	1.710	0.550	
3 Si 32	6.850	6.570	0.280	
3 Si 33	0.459	0.420	0.039	Not used. Made from electrolytic iron that proved to be impure.
3 Si 34	0.915	0.700	0.215	
3 Si 35	0.276	0.193	0.083	
3 Si 36	3.980	3.550	0.430	
3 Si 37	4.810	4.390	0.420	
3 Si 38	2.460	2.260	0.200	Used for mechanical tests only.
3 Si 39	3.560	3.185	0.375	Used for rings for comparative magnetic testing.
3 Si 40	3.560	2.960	0.600	
3 Si 41	2.690	2.360	0.330	Used for mechanical tests only.
3 Si 42	2.690	2.700	−0.010	
3 Si 43	2.770	2.550	0.220	
3 Si 46	2.600	2.530	0.070	
3 Si 47	2.510	2.410	0.100	
3 Si 51	2.660	2.600	0.060	

in the alloys as impurities in a measurable quantity are carbon, amounting to about 0.01 per cent., and oxygen.

It has been previously shown that the electrolytic iron, even after the thorough cleaning to which it was subjected before melting, contained about 0.4 per cent. oxygen in the form of some oxide of iron. It was shown that while carbon will reduce the iron oxide before commencing to combine with the iron, boron will combine with the iron before all the iron oxide is reduced, its affinity for oxygen being about twice its affinity for iron. From Table I it is seen that silicon in this respect acts like boron, with this difference, that its affinity for iron is much stronger than its affinity for oxygen. The percentage of silicon lost, that is, the percentage that has been oxidized and changed into slag, increases, somewhat irregularly, with the silicon added, but reaches a maximum of about 0.5 per cent. As silicon oxidizes at SiO_2, the maximum amount of oxygen absorbed is 0.44 per cent., or the same amount that was found in the two previous investigations, using carbon or boron.

2. *Mechanical Properties*

The results of the mechanical tests are shown in Tables II and III and graphically in Figs. 3 and 4. A comparison between these results

TABLE II.—*Results of Mechanical Tests as Forged*

Number of Specimen	Silicon Content, Per Cent.	Yield Point, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Elongation, Per Cent.		Reduction of Area, Per Cent.	Remarks
				Before "Necking"	Ultimate		
3–39	0.001	35,800	44,700	....	39	80.4	
3–53	0.001	44,400	46,500	3	24	53.8	
3–55	0.001	38,000	40,800	8	40	88.5	
3 Si 16	0.01	41,800	45,200	11	35	78.0	
3 Si 21	0.048	42,850	46,900	5	25	91.6	Failed near punch mark.
3 Si 05	0.068	36,800	43,800	10	37	92.0	
3 Si 22	0.091	35,600	43,750	12	36	91.7	
3 Si 06	0.148	38,600	45,000	11	42	94.8	
3 Si 23	0.205	42,500	49,700	10	39	93.4	
3 Si 17	0.230	41,300	47,500	10	45	89.7	
3 Si 11	0.563	40,750	51,000	12	41	92.6	
3 Si 12	0.673	....	58,000	9	30	91.4	
3 Si 14	0.822	45,200	55,800	11	36	93.1	
3 Si 31	1.71	68,100	76,300	6	29	87.2	
3 Si 38	2.25	66,550	77,750	12	37	82.0	
3 Si 41	2.36	77,200	86,450	8	24	66.0	
3 Si 47	2.41	63,500	75,800	9	23	65.0	
3 Si 46	2.54	60,300	73,400	14	16	20.0	
3 Si 51	2.63	68,700	78,500	13	24	43.0	
3 Si 43	2.65	59,000	74,700	14	34	72.0	
3 Si 42	2.78	68,250	78,000	6	21	74.0	Slight flaw.
3 Si 25	3.40	74,500	86,300	10	31	74.7	
3 Si 36	3.55	83,400	99,300	12	23	41.3	
3 Si 37	4.39	94,000	105,000	6	6	7.5	
3 Si 29	4.92	....	50,250	Nil.	Nil.	Nil.	Failed at base of head.
3 Si 32	6.57	5,120	5,120	Nil.	Nil.	Nil.	Tested between grips without being machined.

and those obtained by some of the previous investigators[10] is given in Figs. 5 and 6. Fig. 7 is a photograph of some of the test pieces after

TABLE III.—*Results of Mechanical Tests Annealed at 970°C.*

Number of Specimen	Silicon Content, Per Cent.	Yield Point, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Elongation, Per Cent.		Reduction of Area, Per Cent.	Remarks
				Before "Necking"	Ultimate		
3–39	0.001	16,400	36,100	....	61	80.9	
3 Si 16	0.010	16,050	34,900	25.0	53	81.5	
3 Si 21	0.048	20,100	35,000	23.0	48	89.3	
3 Si 15	0.064	14,750	34,100	27.0	29	45.1	Failed near head flaw.
3 Si 05	0.068	20,400	34,900	28.0	64	94.8	
3 Si 22	0.091	14,290	35,400	26.0	64	91.8	
3 Si 06	0.148	15,890	35,200	29.0	48	67.0	
3 Si 23	0.205	25,075	38,650	19.0	50	89.4	
3 Si 19	0.230	14,910	35,500	30.0	60	84.7	Broke near head flaw.
3 Si 07	0.242	18,100	38,400	25.0	60	91.3	
3 Si 08	0.309	21,650	40,400	25.0	55	90.0	
3 Si 09	0.400	26,000	42,000	20.0	55	91.0	
3 Si 10	0.472	17,340	42,750	26.0	54	91.4	
3 Si 11	0.563	25,700	41,200	0.08	.....	....	Was not stressed to failure Flaw.
3 Si 12	0.673	26,550	45,230	21.0	45	88.2	
3 Si 13	0.698	23,100	43,000	25.0	57	89.0	
3 Si 14	0.822	26,200	45,150	28.0	50	91.6	
3 Si 31	1.71	35,800	54,250	25.0	50	90.6	
3 Si 18	1.741	45,750	55,000	14.0	......	84.7	Failed at punch mark.
3 Si 38	2.28	42,200	63,500	23.0	50	85.0	
3 Si 43	2.36	46,200	63,600	17.0	29	51.0	
3 Si 41	2.38	43,000	64,200	26.0	50	84.5	
3 Si 47	2.38	47,500	68,700	22.5	45	78.5	
3 Si 46	2.56	47,250	64,200	14.0	18	31.5	
3 Si 51	2.59	42,500	68,200	22.0	42	82.5	
3 Si 42	2.70	48,500	61,800	6.0	6	11.3	
3 Si 27	2.73	49,600	67,800	18.0	19	15.5	
3 Si 25	3.40	57,100	77,400	15.0	21	28.7	
3 Si 37	4.39	85,000	85,000	Nil.	Nil.	1.2	Annealed at 1,030°C. in nitrogen.
3 Si 28	4.44	72,900	91,600	14.0	24	25.1	
3 Si 29	4.92	47,700	47,700	Nil.	Nil.	Nil.	Head failed.
3 Si 32	6.57	13,000	13,000	Nil.	Nil.	Nil.	Tested between grips.
	Annealed at 1,100°C.						
3 Si 41	2.35	39,400	58,300	8.0	10	10.0	
3 Si 47	2.43	41,700	63,100	24.0	45	67.0	
3 Si 46	2.48	45,200	66,300	21.0	47	75.0	
3 Si 43	2.53	43,300	62,500	20.0	40	71.0	
3 Si 42	2.61	38,900	59,500	17.0	32	61.0	Slight flaw.
3 Si 51	2.63	43,300	45,600	1.0	2	Nil.	

being tested, showing very clearly the variation in the elongation and reduction of area caused by silicon.

From the data thus presented it may be stated in general that silicon

[10] Hadfield: *Journal of the Iron and Steel Institute*, vol. xxxvi, p. 222 (1889).
Baker: *Journal of the Iron and Steel Institute*, vol. lxiv, p. 312 (1903).
Guillet: *Revue de Métallurgie*, Memoirs, vol. i, p. 46 (1904).
Paglianti: *Métallurgie*, vol. ix, p. 217 (1912).

increases the strength of iron in almost direct proportion to the amount added, until the maximum strength is reached with a silicon content of about 4.5 per cent. From this point on, the elastic limit coincides with the ultimate strength, and both decrease very rapidly. The limit of forgeability lies between 7 and 8 per cent. The values for the forged condition are considerably higher than for the annealed condition, the difference varying between 10,000 and 20,000 lb. per square inch (7 to 14 kg. per square millimeter). For the 4.5 per cent. alloy "as forged"

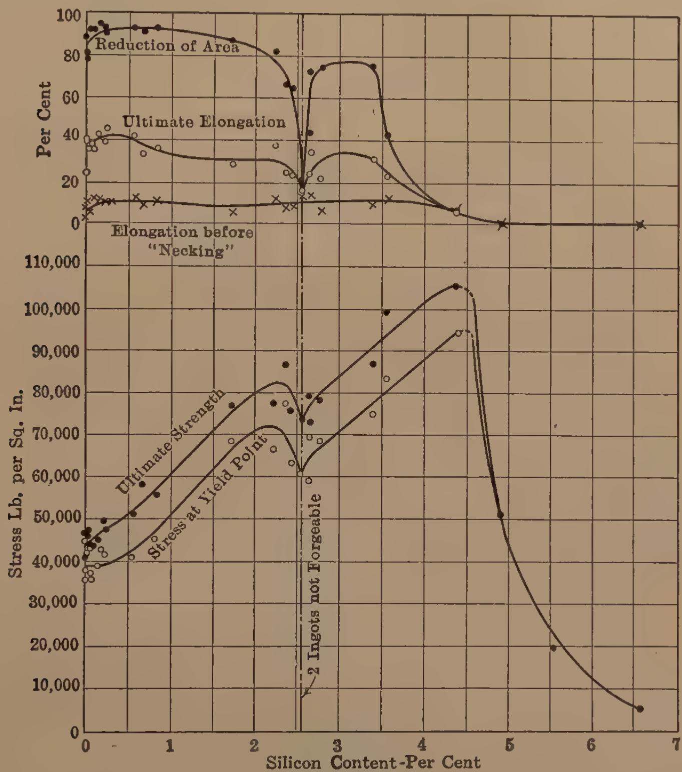

Fig. 3.—Mechanical Properties of Iron-Silicon Alloys. Melted in Vacuo. As Forged.

the ultimate strength is 105,000 lb. per square inch (73.5 kg. per square millimeter), about 8,000 lb. (5.6 kg.) higher than the maximum obtained by previous investigators. The practical absence of carbon in the vacuum iron causes the low-silicon alloys to be weaker than the corresponding alloys tested by previous investigators, but this same absence of carbon is evidently a cause for added strength in the 4.5 per cent. alloy, in which the carbon exists in the form of graphite.

With regard to the elongation and reduction of area, the results in general confirm those obtained by Hadfield, Baker, and Paglianti, con-

cerning the effect of silicon (see Figs. 5 and 6). However, the vacuum alloys again show the effect of the lack of carbon in being much tougher in the region of low silicon as well as in the region between 3 and 5 per cent. The latter is particularly significant, as it is in this region that the maximum strength occurs. While the strength maxima for alloys containing small amounts of carbon correspond to zero elongation and reduction of area, the strongest vacuum alloy, in the forged condition, has a reduction of area of 8 per cent. and an elongation of 7 per cent.

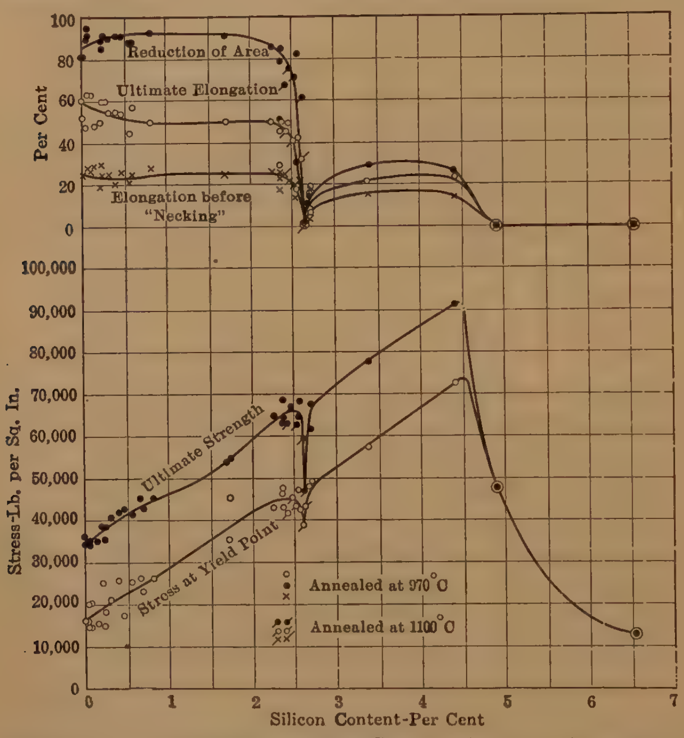

FIG. 4.—MECHANICAL PROPERTIES OF IRON-SILICON ALLOYS. MELTED IN VACUO. ANNEALED.

In the annealed state the corresponding figures for the same alloy are 24 and 22 per cent.

A very interesting feature, as shown by the curves of Figs. 3 and 4, is the critical point that occurs with about 2.60 per cent. silicon, the characteristics of the alloys in this region being their comparative brittleness. This point was first noticed by the fact that two ingots, containing 2.55 and 2.57 per cent. silicon, respectively, were not forgeable but fell into a mass of crystals that apparently had no adhesive strength. One of these alloys is shown in Fig. 15. That this occurrence was no accident is shown by the fact that the two alloys were made at different times, and they were subjected to forging on different days, in company with other

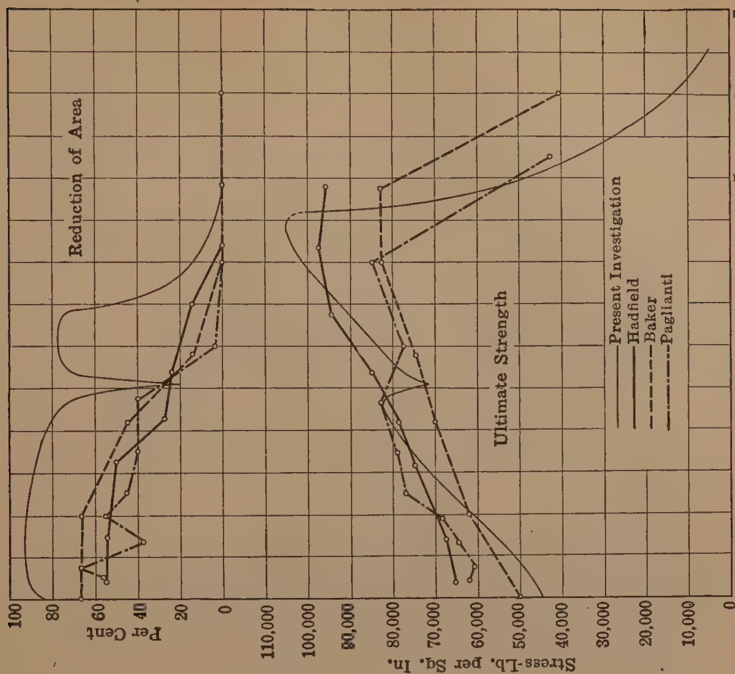

FIG. 5B.—MECHANICAL PROPERTIES OF IRON-SILICON ALLOYS ACCORDING TO VARIOUS INVESTIGATORS. AS FORGED.

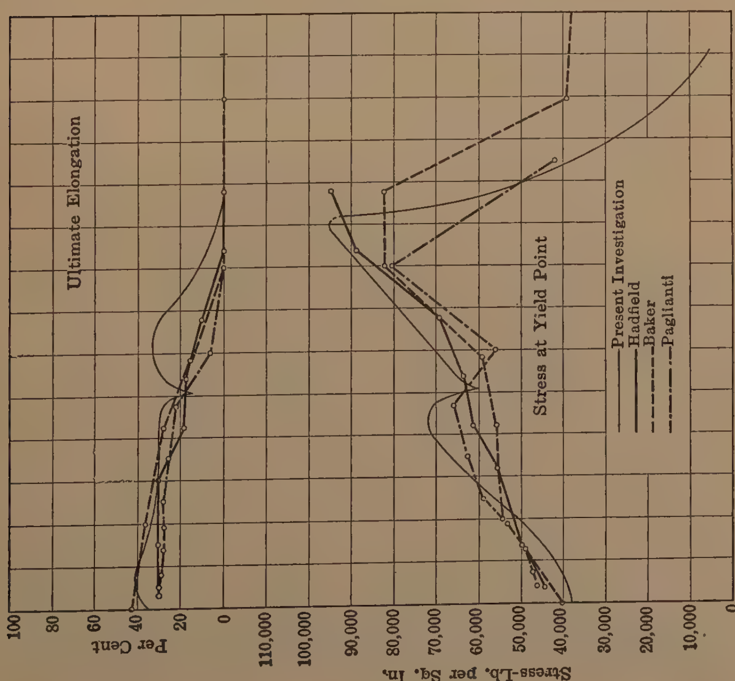

FIG. 5A.—MECHANICAL PROPERTIES OF IRON-SILICON ALLOYS ACCORDING TO VARIOUS INVESTIGATORS. AS FORGED.

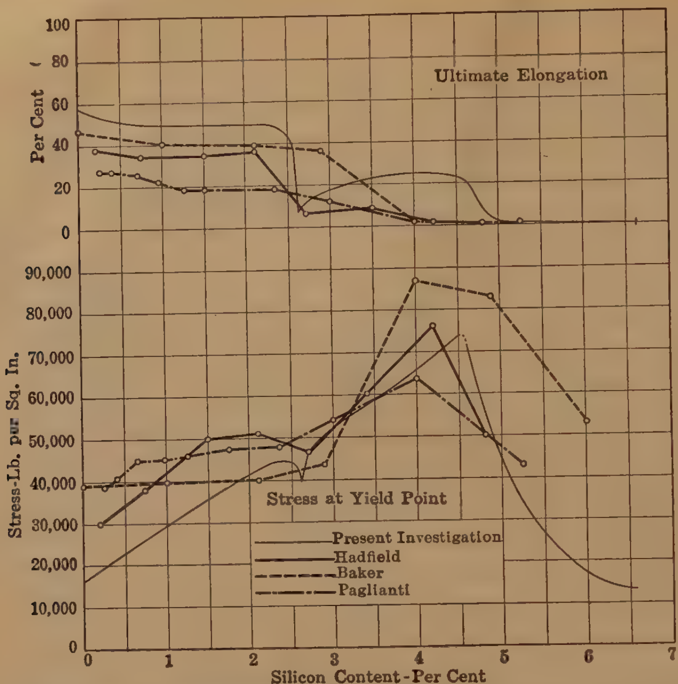

FIG. 6A.—MECHANICAL PROPERTIES OF IRON-SILICON ALLOYS ACCORDING TO VARIOUS INVESTIGATORS. ANNEALED.

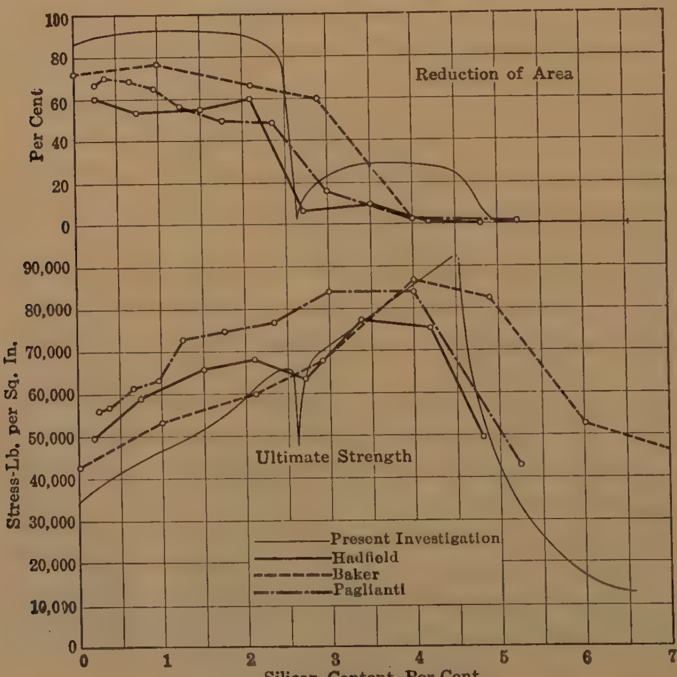

FIG. 6B.—MECHANICAL PROPERTIES OF IRON-SILICON ALLOYS ACCORDING TO VARIOUS INVESTIGATORS. ANNEALED.

alloys that forged perfectly. The structure of the two alloys is identical, consisting of large allotriomorphic crystals ⅛ in. (3 mm.) to ¼ in. (6 mm.) across. Since his first report[11] upon the iron-silicon alloys, where this critical point was mentioned, the author has obtained additional experimental data in the region in which this point occurs. These data

Fig. 7.—Some of the Mechanical Test Pieces After Being Tested.

are included in the present report, making it possible to draw definite curves through the critical range. The critical point appears in all the characteristic curves for the mechanical properties both for the unannealed and for the annealed alloys. For the unannealed specimens

[11] *Proceedings of the American Institute of Electrical Engineers*, vol. xxxiv, p. 2455 (October, 1915).

there is a sudden drop in the reduction of area and ultimate elongation at 2.25 per cent. silicon, the curves reaching their minima at about 2.56 per cent., which is the silicon content of the two non-forgeable alloys. With increasing silicon contents these curves again rise as suddenly as they previously dropped, attaining about the same values as before the critical point was reached. At 3.50 per cent. the second and final drop begins for these curves, reaching zero at about 5 per cent. silicon.

For the annealed specimens the critical point is even more marked, but the reduction of area and the elongation do not recover as completely after the critical point is passed as was the case with the unannealed specimens. The critical point for the annealed alloys occurs with a silicon content of about 2.65 per cent., slightly higher than for the unannealed alloys.

TABLE IV.—*Results of Magnetic and Electrical Tests, Rods Annealed at* 900°C.

Rod No.	Silicon Content, Per Cent.	Maximum Permeability	Density for Max. Permeability, Gausses	Permeability for $B = 10,000$	Hysteresis Loss, Ergs / C.c./ Cycle		Retentivity, Gausses		Coercive Force, Gilberts / Cm.		Spec. Elec. Res. at 20°C., Microhms	Remarks
					For $B_{max} = 10,000$	For $B_{max} = 15,000$	For $B_{max} = 10,000$	For $B_{max} = 15,000$	For $B_{max} = 10,000$	For $B_{max} = 15,000$		
3–54	0.001	23,100	8,500	21,800	764.0	1,610.0	9,400	14,200	0.25	0.30	9.85	
3–55	0.001	22,500	11,000	21,300	875.0	1,790.0	9,100	13,800	0.29	0.36	9.82	
3 Si 16	0.010	25,000	10,000	25,000	795.0	1,770.0	9,480	14,600	0.28	.0.35	9.89	
3 Si 15	0.064	22,800	10,000	21,700	782.0	1,738.0	9,100	14,500	0.26	0.35	10.65	
3 Si 05	0.068	37,500	9,000	36,300	405.0	1,210.5	9,480	14,520	0.12	0.23	10.75	
3 Si 06	0.148	47,000	8,000	42,500	396.3	965.0	9,300	14,100	0.12	0.19	11.80	
3 Si 17	0.230	30,000	6,000	26,300	496.0	1,311.0	9,000	13,600	0.15	0.23	11.64	
3 Si 10	0.472	14,000	7,000	12,700	960.0	1,863.0	8,900	11,700	0.30	0.33	16.20	Rod strained in testing.
3 Si 14	0.822	13,500	9,000	13,300	1,215.0	2,432.0	9,100	12,400	0.42	0.53	21.3	Rod strained in testing.
3 Si 31	1.71	18,000	7,000	15,900	800.0	1,541.0	8,400	11,540	0.26	0.32	33.2	
3 Si 18	1.741	14,300	8,000	14,100	935.0	2,162.0	9,000	12,600	0.29	0.42	31.2	
3 Si 27	2.73	16,800	6,000	13,300	821.0	1,779.0	8,270	10,500	0.27	0.34	41.8	
3 Si 25	3.40	20,000	6,000	15,900	560.0	1,390.0	7,900	9,570	0.20	0.23	48.5	
3 Si 36	3.55	14,000	4,500	8,850	802.5	1,812.0	6,900	8,100	0.28	0.34	51.5	Rods contaminated by impure rods.
3 Si 37	4.39	12,750	4,500	7,500	846.0	1,956.0	6,600	7,700	0.29	0.31	58.8	
3 Si 28	4.44	16,100	4,800	10,100	623.0	1,575.0	7,000	8,370	0.16	0.24	57.7	
3 Si 29	4.92	9,100	4,500	5,330	776.0	2,006.0	4,500	5,300	0.27	0.36	66.5	

The author knows of no satisfactory explanation of this critical point, nor has he been able to find any such phenomenon reported by anybody else. From Figs. 5 and 6 it is seen, however, that both Hadfield and Paglianti, and to some extent, Baker, indicate irregularities in some of their curves for a silicon content between 2 and 3 per cent. None of the above investigators, however, and, as far as the author has been able to find out, no other investigator, has recorded in the literature the properties of an iron-silicon alloy containing between 2.50 and 2.67 per cent. silicon.

As critical points are usually associated with the formation of definite compounds of the elements present, it may be of some interest to note that a compound of the formula $Fe_{19}Si$ would contain 2.56 per cent. silicon, and similarly that a compound of the formula $Fe_{19}Si_2$ would contain 4.99 per cent. silicon. It was stated above that the first critical point in the present case occurs with a silicon content of 2.56 per cent., and also that there is a sudden change at about 5 per cent. silicon. Whether this agreement is a mere coincidence, or whether these compounds, or others, actually exist, the author is not prepared to say, as conclusive evidence in the way of cooling curves for these particular alloys is not available.

TABLE V.—*Results of Magnetic and Electrical Tests, Rods Annealed at* 1,100°*C.*

Rod No.	Silicon Content, Per Cent.	Maximum Permeability	Density for Max. Permeability, Gausses	Permeability for $B = 10,000$	Hysteresis Loss, Ergs / C.c. / Cycle		Retentivity, Gausses		Coercive Force, Gilberts / Cm.		Spec. Elec. Res. at 20°C., Microhms	Remarks
					For $B_{max} = 10,000$	For $B_{max} = 15,000$	For $B_{max} = 10,000$	For $B_{max} = 15,000$	For $B_{max} = 10,000$	For $B_{max} = 15,000$		
3–54	0.001	22,800	8,000	21,300	665.0	1,860.0	9,300	13,300	0.20	0.24	9.84	
3–55	0.001	25,800	9,000	25,600	707.0	1,451.0	9,300	12,700	0.23	0.28	9.85	
3 Si 16	0.010	29,000	9,000	28,670	707.0	1,604.0	9,600	14,300	0.21	0.31	9.9	
3 Si 21	0.048	27,000	10,000	27,000	700.0	1,660.0	9,440	14,480	0.23	0.32	10.5	
3 Si 15	0.064	36,800	9,000	36,300	502.5	1,336.0	9,500	14,300	0.16	0.25	10.67	
3 Si 05	0.068	44,200	9,000	43,500	407.0	1,214.5	9,480	14,200	0.13	0.225	10.78	
3 Si 22	0.091	45,250	9,000	43,500	394.0	929.0	9,500	14,300	0.13	0.17	10.96	
3 Si 06	0.148	66,500	6,500	41,700	286.0	916.0	9,080	12,000	0.09	0.165	11.80	
3 Si 23	0.205	30,200	9,000	29,500	649.0	1,526.0	9,300	14,480	0.20	0.27	12.5	
3 Si 17	0.230	......	......	......	......	......	.....	......	.....	.....	.....	Contaminated by impure rods.
3 Si 07	0.242	36,500	7,500	33,000	436.0	1,346.0	9,700	14,500	0.13	0.21	13.4	
3 Si 08	0.309	44,800	9,000	43,500	445.0	1,412.0	9,600	14,500	0.13	0.24	14.4	
3 Si 09	0.400	22,500	9,000	22,000	725.0	1,820.0	9,440	14,480	0.21	0.32	15.3	
3 Si 10	0.472	31,150	6,200	25,000	535.0	1,358.0	9,300	14,200	0.16	0.21	16.57	
3 Si 11	0.563	25,000	9,000	25,000	601.5	1,624.0	9,200	14,320	0.20	0.28	17.50	
3 Si 12	0.673	28,000	7,000	24,500	468.0	1,636.0	9,200	13,670	0.13	0.23	19.10	
3 Si 13	0.698	20,350	8,000	19,600	780.0	2,220.0	9,300	14,400	0.25	0.40	19.60	
3 Si 14	0.822	30,800	9,500	30,300	542.0	1,765.0	9,200	14,100	0.18	0.35	21.25	
3 Si 31	1.710	30,150	6,500	24,700	440.0	1,292.0	8,700	12,000	0.12	0.22	33.25	
3 Si 18	1.741	33,000	7,000	26,300	416.0	1,112.0	9,200	12,600	0.13	0.19	31.00	
3 Si 27	2.730	46,800	9,500	46,000	404.0	1,260.0	9,100	13,300	0.13	0.23	42.00	
3 Si 25	3.400	63,300	6,500	46,500	280.0	1,025.0	9,100	12,400	0.08	0.15	48.50	
3 Si 36	3.550	36,000	7,500	29,500	419.0	1,157.0	8,920	12,000	0.13	0.21	48.50	Rods annealed in nitrogen.
3 Si 37	4.390	25,700	6,000	15,400	591.0	1,819.0	8,300	10,200	0.20	0.25	56.10	
3 Si 28	4.440	30,200	3,000	15,900	405.0	1,171.0	7,000	8,000	0.12	0.15	57.40	
3 Si 29	4.920	12,200	5,000	7,040	780.0	2,620.0	6,300	7,100	0.26	0.35	66.20	

3. *Magnetic and Electrical Properties*

The results of the magnetic and electrical tests are shown in Tables IV and V and in Figs. 8 and 9. The latter show at a glance the magnetic and electrical properties of the series, Fig. 8 after annealing at 900°C.,

and Fig. 9 after annealing at 1,100°C. Two maxima appear very distinctly in the curves for maximum permeability, corresponding to two minima in the curves for hysteresis loss and coercive force. The first of these points occurs at a silicon content of about 0.15 per cent., and the second at a silicon content of about 3.5 per cent.

That a maximum—or minimum—should occur for a low silicon content was not surprising in view of the results previously obtained with

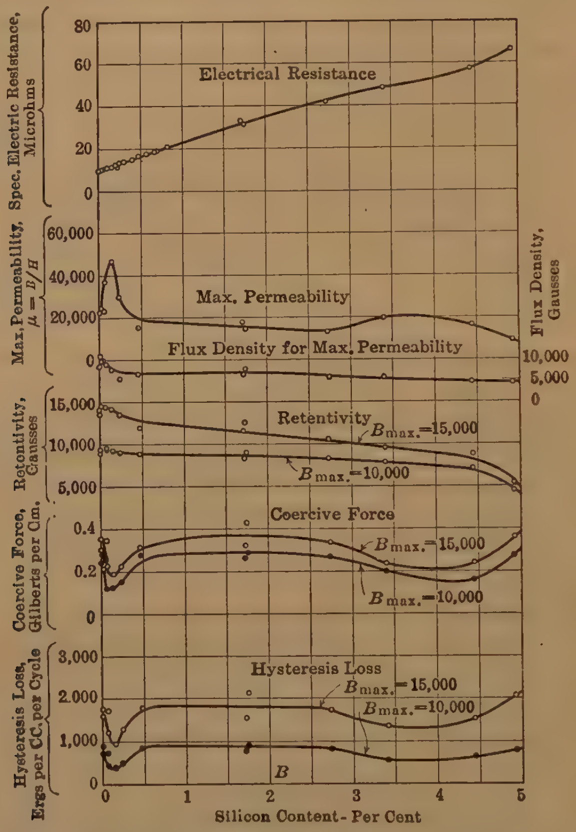

FIG. 8.—MAGNETIC AND ELECTRICAL PROPERTIES OF IRON-SILICON ALLOYS. MELTED IN VACUO. ANNEALED AT 900°C.

pure iron, iron-carbon and iron-boron alloys. In the latter case a maximum was obtained with a trace of boron, evidently on account of a slight purification of the iron, but as soon as the boron content became measurable, the magnetic properties immediately depreciated. The first maximum in the present case can, no doubt, be accounted for in the same way, and consequently this point may be regarded as characteristic of the purest iron obtainable under the present conditions, containing 0.15 per cent. silicon and a small amount of oxygen in the form of iron oxide.

The second maximum—or minimum—was wholly unexpected, as strength and brittleness are not generally associated with high magnetic quality. It is true that previous investigators[12] have found a maximum between 2.5 and 4.0 per cent. silicon, but this was thought to be due to

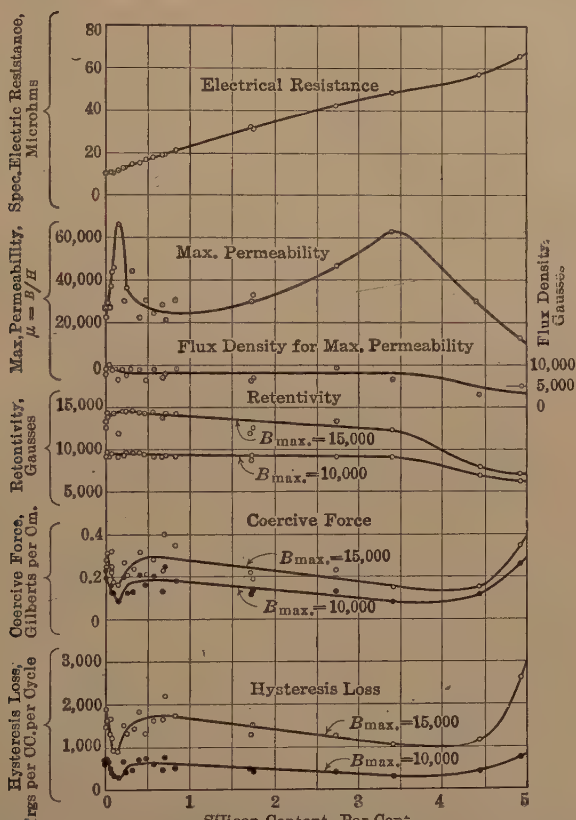

FIG. 9.—MAGNETIC AND ELECTRICAL PROPERTIES OF IRON ALLOYS. MELTED IN VACUO. ANNEALED AT 1,100°C.

the neutralizing effect of the silicon upon the relatively large amounts of impurities, chiefly carbon, present in the iron. In the present case

[12] Barrett, Brown, and Hadfield: *Scientific Transactions of the Royal Dublin Society*, vol. vii, Series II, Part IV, p. 67 (January, 1900). *Journal of the Institution of Electrical Engineers*, vol. xxxi, p. 674 (1901–02).

Gumlich and Schmidt: *Elektrotechnische Zeitschrift*, vol. xxii, p. 691 (1901).

Baker: *Journal of the Iron and Steel Institute*, vol. lxiv, p. 312 (1903); *Journal of the Institution of Electrical Engineers*, vol. xxxiv, p. 498 (1904–05).

Burgess and Aston: *Metallurgical and Chemical Engineering*, vol. viii, p. 131 (March, 1910).

Gumlich and Goerens: *Transactions of the Faraday Society*, vol. viii, p. 98 (1912).

Paglianti: *Métallurgie*, vol. ix, p. 217 (1912).

the alloys contained only about 0.01 per cent. of carbon. Thus it seems improbable that the second maximum in this case can be attributed solely to the conversion of 0.01 per cent. of combined carbon into graphite. It seems more probable that the improvements are due partly to this conversion and partly to the complete reduction of iron oxide. According to those hypotheses, the first maximum is due to pure iron in spite of small amounts of iron oxide and combined carbon, while the second maximum is due to pure iron in spite of a relatively large amount of dissolved silicon. If none of the above hypotheses is correct, the only other explanation remaining is that the second maximum is due directly to silicon dissolved in the iron. As an argument against such a theory, Hadfield and Hopkinson[13] in 1900, and Gumlich[14] in 1912, brought out the fact that silicon reduces the saturation value of iron in direct proportion to the silicon dissolved in the iron, and consequently it did not seem probable that silicon could directly improve the permeability at lower densities. However, it is a curious coincidence that at the same meeting of the Faraday Society at which Dr. Gumlich made the above statement, Dr. P. Weiss[15] read a paper on iron-cobalt alloys, showing that the alloy Fe_2Co has a saturation value 10 per cent. higher than that of pure iron. The author,[16] in coöperation with Dr. E. H. Williams,[17] has shown that while the iron-cobalt alloy Fe_2Co, melted *in vacuo*, has a saturation value 13 per cent. higher than that of pure iron melted under identical conditions, its permeability at low densities is much lower. Evidently both the high saturation value and the comparatively low permeability at low densities of the Fe_2Co alloy must be due to the combination between iron and cobalt, or, in other words, must be attributed directly to the cobalt. There is no reason, then, why the low saturation value and the high permeability at low densities of the 3.40 per cent. iron-silicon alloy cannot both be due directly to silicon. That is, no foundation exists any longer for assuming that there is a direct connection between the saturation value of a certain alloy and its properties at low and medium densities, and it is consequently possible that the second maximum occurring in the maximum permeability curve for the iron-silicon series may be due directly to silicon. More experimental evidence is needed, however, before it is safe to make a definite statement in this respect.

While silicon is thus, directly or indirectly, engaged in improving the magnetic properties of iron, it serves a very useful purpose by enor-

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[13] *Journal of the Institution of Electrical Engineers*, vol. xlvi, p. 225 (1911).

[14] *Transactions of the Faraday Society*, vol. viii, p. 109 (1912).

[15] *Transactions of the Faraday Society*, vol. viii, p. 149 (1912).

[16] *General Electric Review*, vol. xviii, p. 881 (September, 1915).

Proceedings of the American Institute of Electrical Engineers, vol. xxxiv, No. 10, p. 2463 (October, 1915).

[17] *Physical Review*, vol. vi, p. 404 (1915).

mously increasing the electrical resistance of iron, giving the iron the exact characteristics desired for electromagnetic machinery.

The iron-silicon series thus offers two important alloys for electrical purposes, both having high permeability and low hysteresis loss, but differing in that one has a very low, while the other has a very high, electrical resistance.

The values obtained are, undoubtedly, without precedent in the annals

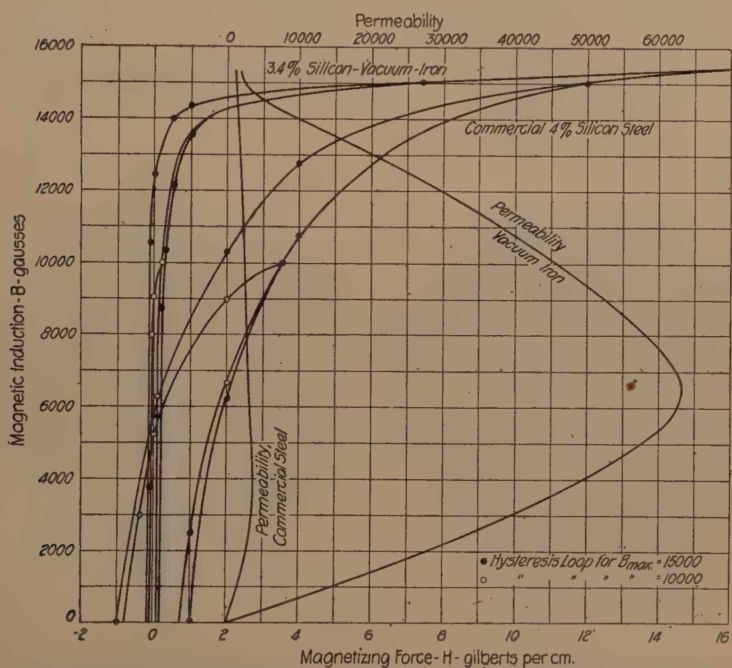


FIG. 10.—COMPARISON BETWEEN VACUUM IRON AND COMMERCIAL STEEL, BOTH CONTAINING BETWEEN 3 AND 4 PER CENT. SILICON. THOROUGHLY ANNEALED.

	Vacuum Iron	Commercial Steel
Hysteresis Loss for $B_{max.}$ = 10,000-ergs. per c.c. per cycle.	280.00	2160.00
Hysteresis Loss for $B_{max.}$ = 15,000-ergs. per c.c. per cycle.	1025.00	4290.00
Specified Electrical Resistance-microhms.	48.50	51.15

of the magnetic properties of iron and iron alloys, and it is only after a careful analysis of the apparatus used and the methods employed in testing that the author feels justified in publishing them. As an extra precaution, however, he wishes to repeat the statement made under "Magnetic Testing" that experimental evidence seems to point toward a larger percentage error due to the compensating current than theoretical con-

siderations would lead to.[18] But even if the maximum error in the results as given should amount to 20 per cent., their significance would not be altered appreciably. Whether the true maximum permeability obtained is 66,500 or 53,200 is of little consequence at the present time, as long as it is reasonably certain that it lies in this neighborhood.

Fig. 10 illustrates the difference in magnetic quality between a silicon-vacuum iron and a commercial silicon steel, both containing approximately the same amount of silicon. The maximum permeability is as 20 to 1, the hysteresis loss, for B_{max} = 10,000, as 8 to 1, and for B_{max} = 15,000, as 4 to 1, in favor of the vacuum alloy. For B_{max} = 15,000 the permeability of the latter is twice the permeability of the commercial iron. The electrical resistance is nearly the same for both alloys, or about five times that of pure iron.

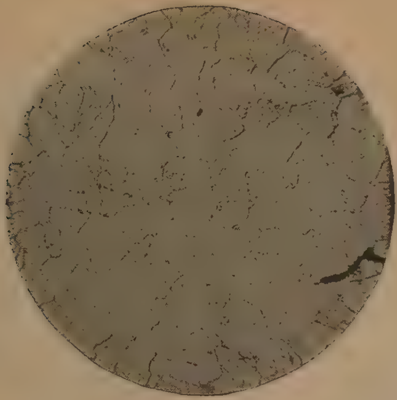
4. *Photomicrographs*

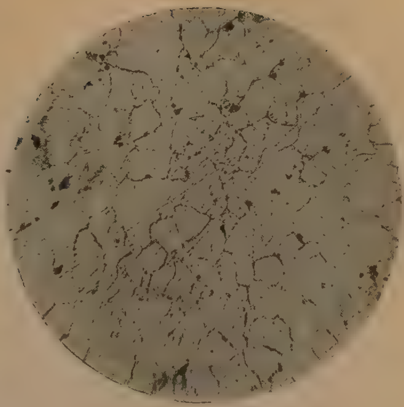
In the following pages a number of photomicrographs representative of the alloys tested are reproduced. They are arranged in order of their silicon content, the pure iron appearing first and the highest silicon alloy last. With only a few exceptions the "as forged" condition occupies the upper part of the pages, the 900° annealed condition the middle part, and the 1,100° annealed condition the lower part. The magnifications used are either 40 diameters or 10 diameters. In one case recourse was had to 7 diameters. Even with the information thus at hand, exact definite conclusions cannot be made with regard to each alloy taken separately, as the specimen may not in every case be strictly representative of the particular alloy from which it was taken. Furthermore, in comparing the photomicrographs after the different heat treatments, it should be remembered that the specimens had to be repolished after each treatment, and the photomicrographs do not, therefore, represent the same spot in each case. However, general conclusions may be drawn by considering the series as a whole.

From these photomicrographs it is seen very clearly that the iron-silicon alloys, in the range here investigated, consist of only one kind of crystals, thus confirming the results obtained by previous investigators[19]

[18] While the experiments in regard to this matter are not yet complete, it is safe to say that the values as given in the paper should be modified to a certain extent. Thus the permeabilities as given are generally too high, as are also the values for residual magnetism and coercive force. While the hysteresis loss for B_{max} = 10,000 is too low, the loss for B_{max} = 15,000 is too high.

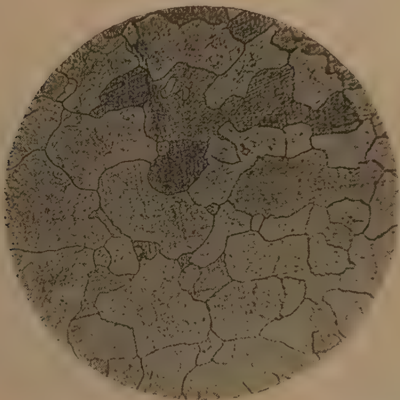
[19] Osmond: *Journal of the Iron and Steel Institute*, vol. xxxvii, p. 62 (1890).
Arnold: *Journal of the Iron and Steel Institute*, vol. xlv, p. 107 (1894).
Baker: *Journal of the Iron and Steel Institute*, vol. lxiv, p. 312 (1903).
Guertler and Tammann: *Zeitschrift für anorganische chemie*, vol. xlvii, p. 165 (1905); vol. lix, p. 384 (1908).
Gontermann: *Journal of the Iron and Steel Institute*, vol. lxxxiii, p. 431 (1911).

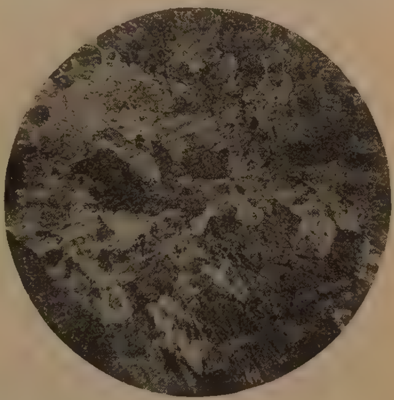




As Forged.

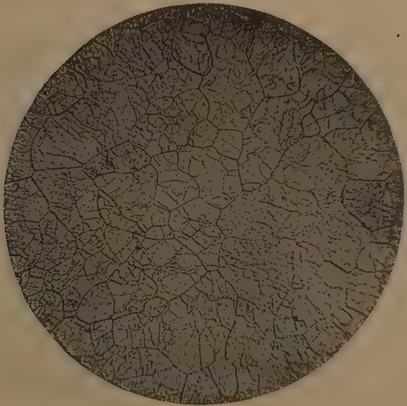
FIG. 11A.—× 40 DIAM. PICRIC ACID. FIG. 12A.—× 40 DIAM. PICRIC ACID.

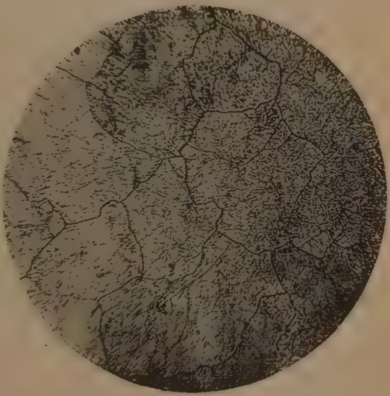




Annealed at 900°.

FIG. 11B.—× 40 DIAM. PICRIC ACID. FIG. 12B.—× 40 DIAM. PICRIC ACID.



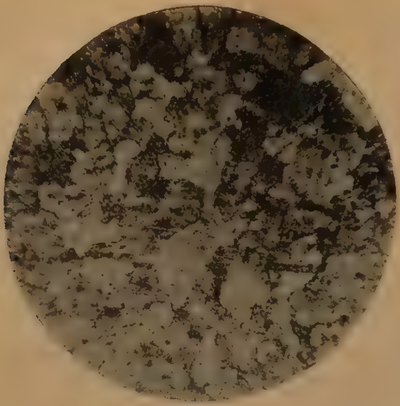


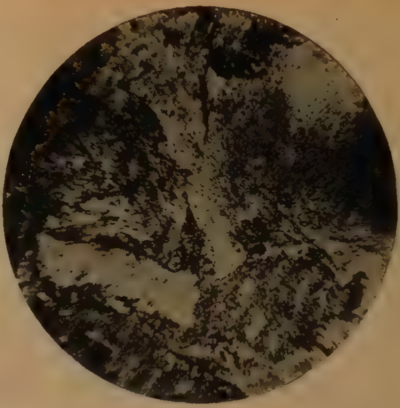
Annealed at 1100°.

FIG. 11C.—× 40 Diam. PICRIC ACID. FIG. 12C.—× 40 DIAM. PICRIC ACID.

ALLOY NO. 3–39. 0.001 PER CENT. SILICON.

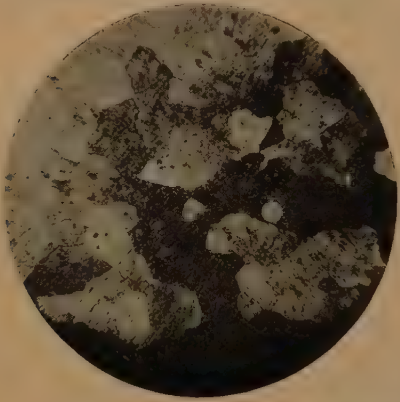
ALLOY NO. 3 SI 06. 0.148 PER CENT. SILICON.

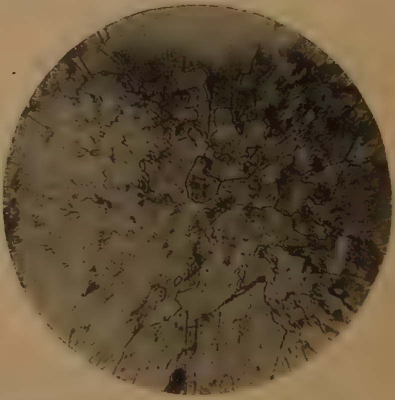




As Forged.

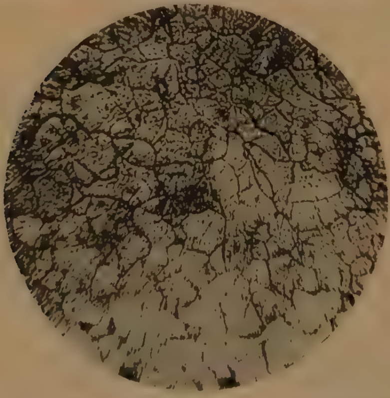
FIG. 13A.—× 40 DIAM. PICRIC ACID. FIG. 14A.—× 10 DIAM. PICRIC ACID.

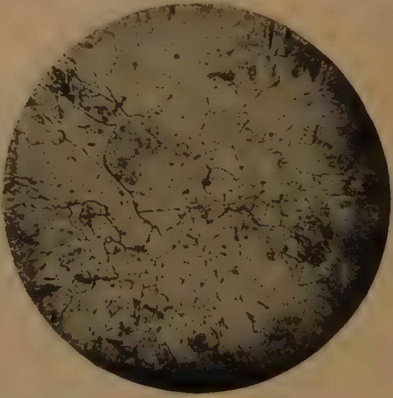




Annealed at 900°.

FIG. 13B.—× 40 DIAM. PICRIC ACID. FIG. 14B.—× 10 DIAM. PICRIC ACID.



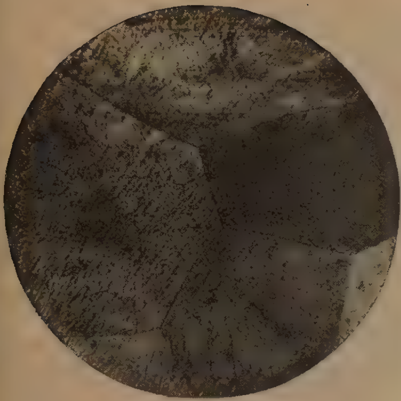


Annealed at 1000°.

FIG. 13C.—× 40 DIAM. PICRIC ACID. FIG. 14C.—× 10 DIAM. PICRIC ACID.

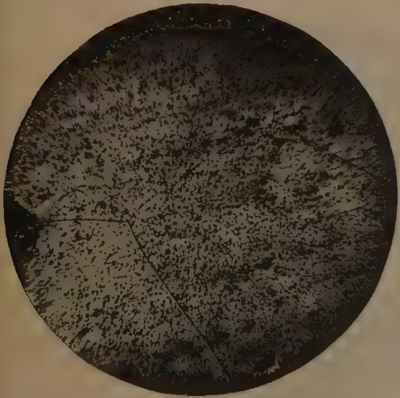
ALLOY NO. 3 Si 10. 0.472 PER CENT. SILICON.

ALLOY NO. 3 Si 31. 1.71 PER CENT. SILICON.



As Cut from Ingot Before Forging.
FIG. 15A.—× 10 DIAM. PICRIC ACID

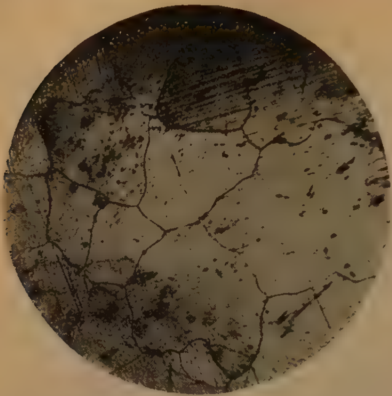
Annealed at 900°.
FIG. 15B.—× 10 DIAM. PICRIC ACID.

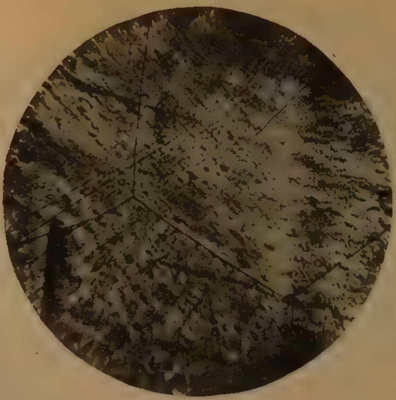


Annealed at 1100°.
FIG. 15C.—× 10 DIAM. PICRIC ACID.

FIG. 15A2.—NEARLY FULL-SIZE INGOT AFTER FORGING.

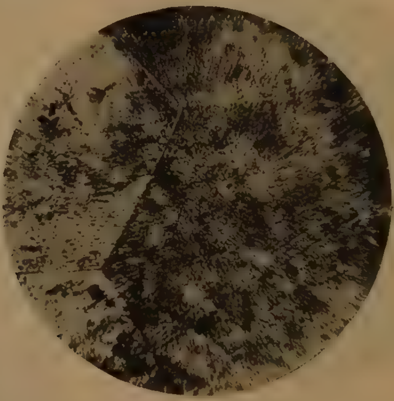
NON-FORGEABLE ALLOY NO. 3 SI 24
2.57 PER CENT. SILICON.





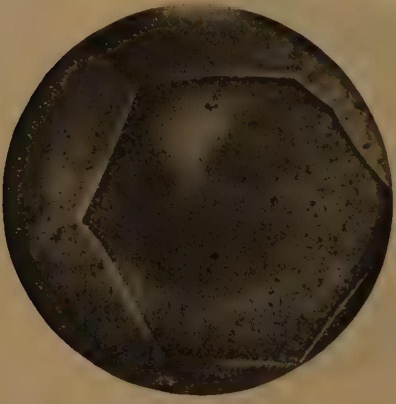
As Forged.

FIG. 16.—× 40 DIAM. NITRIC ACID. FIG. 17A.—× 10 DIAM. PICRIC ACID.



Annealed at 900°.

FIG. 17B.—× 10 DIAM. PICRIC ACID.

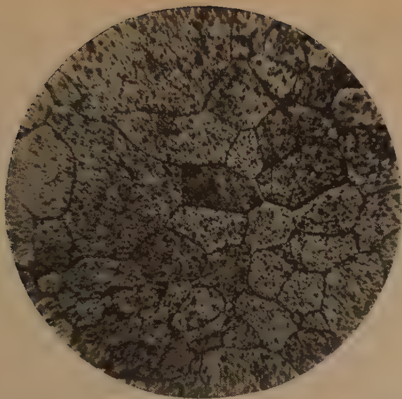


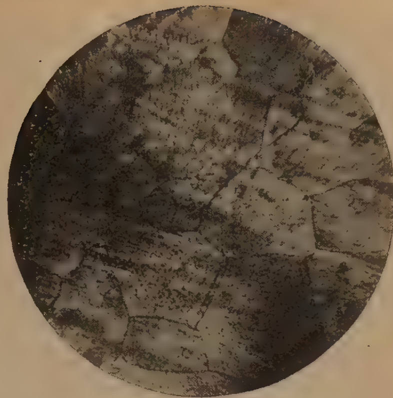
Annealed at 1100°.

FIG. 17C.—× 7 DIAM. PICRIC ACID.

SPEC. 1. SPEC. 2.

ALLOY NO. 3 SI 25. 3.40 PER CENT. SILICON.

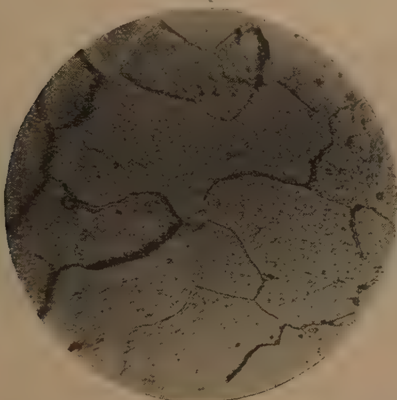


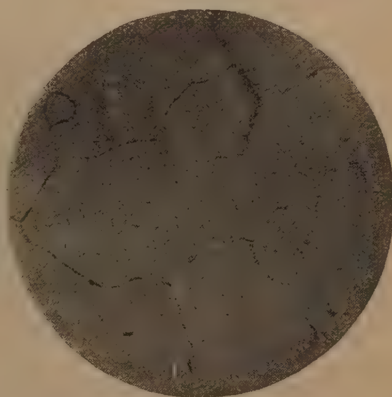


As Forged.

FIG. 18A.—× 40 DIAM. PICRIC ACID.

FIG. 19A.—× 40 DIAM. PICRIC ACID.

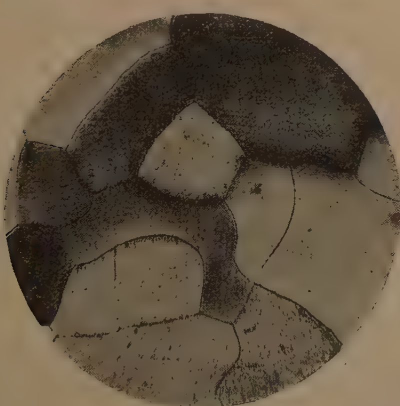


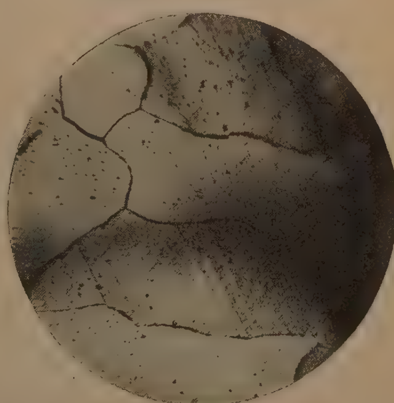


Annealed at 900°.

FIG. 18B.—× 40 DIAM. PICRIC ACID.

FIG. 19B.—× 40 DIAM. PICRIC ACID. SLIGHTLY REPOLISHED.





Annealed at 1100°.

FIG. 18C.—× 40 DIAM. PICRIC ACID. WASHED IN 10 PER CENT. HYDROFLUORIC SOLUTION.

FIG. 19C.—× 40 DIAM. PICRIC ACID. WASHED IN 10 PER CENT. HYDROFLUORIC SOLUTION.

ALLOY NO. 3 SI 29. 4.92 PER CENT. SILICON.

ALLOY NO. 3 SI 32. 6.57 PER CENT. SILICON.

that iron and silicon, for silicon contents below about 15 per cent., form a solid solution everywhere between the freezing point and ordinary temperature.

Below 1 per cent., silicon appears to have no marked effect upon the structure of the iron, either as forged or annealed. Annealing at 900°C. does not seem to change the structure of the alloys in this range, but annealing at 1,100° breaks up the large crystals into smaller ones, giving an appearance of a very fine structure.[20] There is no sign of foreign substances in these structures, other than those arising from imperfect polishing. After passing the 1 per cent. mark, silicon begins to show its effect. The crystals are generally larger than for the pure iron and are readily polished in relief, showing that they are not of uniform hardness. There is no breaking up of the crystals by the 1,100° annealing as in the case of the low alloys. The 1.71 per cent. alloy, as seen in Fig. 14, shows

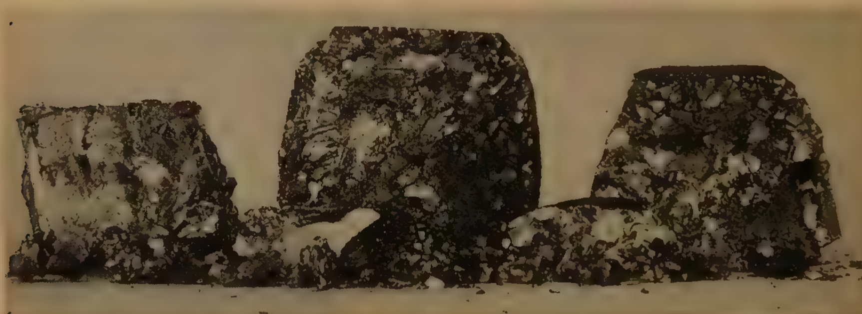


FIG. 20.—NON-FORGEABLE ALLOY NO. 3 SI 30. 8.55 PER CENT. SILICON. INGOT AFTER FORGING. NEARLY HALF-SIZE.

a very irregular structure as forged, and after the 900° annealing, but this gives way to a structure of more regularity by the 1,100° annealing. The structure of the non-forgeable alloy is shown in Fig. 15, exhibiting very large, uniform crystals, measuring ⅛ in. (3 mm.) to ¼ in. (6 mm.) across. In the first specimen used for the 3.40 per cent. alloy, the crystals are of about the same size and shape as the ones for the non-forgeable alloy, as seen in Fig. 17. In order to further investigate this singularity, another specimen from the same alloy was prepared, and this showed a much more normal structure (Fig. 16). Specimens for two other alloys

[20] This phenomenon is explained by Stead and Carpenter, *Journal of the Iron and Steel Institute* (September, 1913) as follows: Upon heating the iron above the Ar_3 point (900°C. for pure iron) and holding it in the region of the gamma modification sufficiently long, both the alpha crystals and their nuclei will be destroyed, giving place to gamma crystals. If now the iron is cooled numerous alpha nuclei will be formed simultaneously on passing through Ar_3, and the resulting crystals are consequently small.

See also Oliver W. Stovey: A Microscopic Study of Electrolytic Iron, *Transactions of the American Electrochemical Society*, vol. v, p. 201 (1904).

with very nearly the same silicon content were also investigated, and these also showed structures that were quite normal, so that the large crystals, shown in Fig. 17, are evidently freaks, caused by peculiar conditions. It may be that the specimen was taken at one end of the forged rod. Nevertheless, the occurrence of these enormous crystals is very interesting, not only on account of the enormous size, but also because it shows that the size of the crystals in and of itself does not prevent the material from being forgeable. The remainder of the photomicrographs exhibit quite normal structures with no marked change caused by the two heat treatments.

The principal difference between these photomicrographs and the ones published by previous investigators[21] is the absence of foreign matter in the structure of the vacuum alloys. Even Baker, whose alloys contain only 0.04 per cent. carbon, shows in his photomicrographs besides small amounts of pearlite or graphite, some other foreign substance that could not be explained at the time. It seems probable, in view of the method used for melting and the mechanical properties of his alloys, that these other foreign substances may be oxides. The other investigators invariably show comparatively large amounts of pearlite for low alloys and patches or spots of graphite for high alloys. The size of crystals for the vacuum alloys, excluding the abnormal cases, is very much larger than for the less pure alloys. This is true both for low and high silicon contents.

IV. Summary and Conclusions

The results recorded in the previous pages may be summarized as follows:

1. By means of the vacuum method of melting it is possible to obtain a decidedly purer product than has thus far been obtained in any other manner. Consequently by the use of this method more definite conclusions, than have hitherto been possible, can be drawn with regard to the effect of silicon upon iron.

2. Silicon, like boron, has a double effect upon iron. Part of it combines with the iron and remains in solid solution throughout the cooling of the alloy, while a smaller part reduces the iron oxide present.

3. The tensile strength of the vacuum product follows in general the same law as alloys made under ordinary conditions, but the ductility of the former is much greater, particularly below 2 per cent. and above 3 per cent., probably due to the absence of carbon. The maximum tensile strength of 105,000 lb. per square inch (73.5 kg. per square millimeter) occurs with a silicon content of 4.5 per cent.

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[21] Guillet: *Revue de Métallurgie*, vol. i, p. 46 (1904).
Gumlich and Goerens: *Transactions of the Faraday Society*, vol. viii, p. 98 (1912).
Paglianti: *Métallurgie*, vol. ix, p. 217 (1912).

4. The limit of forgeability lies between 7 and 8 per cent. silicon. A critica' range occurs between 2.50 and 2.60 per cent., in which the alloys are exceedingly brittle, in some cases being not even forgeable. The chemical composition of the latter corresponds to a compound of the formula $Fe_{19}Si$.

5. With regard to the magnet c properties, the vacuum alloys exhibit most remarkable characteristics. The best alloys are obtained with about 0.15 per cent. and 3.40 per cent silicon after annealing at 1,100°C. The maximum permeability for both of these alloys is above 50,000, and the hysteresis loss for B_{max} = 10,000 and 15,000 is about 300 and 1,000 ergs per cubic centimeter per cycle respectively. This hysteresis loss is one-eighth and one-fourth of the corresponding loss for commercial silicon steel. The most favorable annealing temperature is in every case 1,100°C.

6. The specific electrical resistance increases about 13 microhms for the first per cent. silicon added. For each additional per cent. added the increase is about 11 microhms. Consequently the 3.40 per cent. alloy mentioned under 5 has a resistance nearly five times that of the 0.15 per cent. alloy.

By the vacuum process two silicon alloys have thus been produced that have very valuable characteristics; one, low in silicon, not very strong, but extremely ductile, of high permeability, low hysteresis loss, and of low electrical resistance; the other high in silicon, very strong, moderately tough, of high permeability, low hysteresis loss and of high electrical resistance. The properties for these two alloys are summarized in Table VI. The first is evidently suitable for use in places where high permeability and low hysteresis loss are the chief requirements, while the second alloy is suitable for electromagnetic machinery, principally transformers, where a low eddy-current loss is an additional requirement.

TABLE VI.—*Properties of the Two Best Iron-Silicon Vacuum Alloys*

Silicon Content, Per Cent.	Stress at Yield Point, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Elongation, Per Cent.	Reduction of Area, Per Cent.	Maximum Permeability	Density for Max. Permeability, Gausses	Hysteresis Loss, Ergs/c.c./Cycle		Spec. Elect. Resistance, Microhms
							For B_{max} = 10,000	For B_{max} = 15,000	
0.15	18,500	37,000	56	90.0	above 50,000	6,500	286[1]	916[2]	11.80
3.40	58,000	76,500	21	28.5	above 50,000	6,500	280[1]	1,025[2]	48.50

[1] From data recently obtained with rings these values may be 10 to 20 per cent. low.
[2] From data recently obtained with rings these values may be 10 to 20 per cent. high.

The mechanical properties of the second alloy make possible its use also in dynamo machinery, where the present commercial silicon steel cannot be used on account of its brittleness.

It should be pointed out that electrolytic iron is not essential to the attainment of high magnetic quality. Any low-carbon iron that is

practically free from phosphorus, sulphur and manganese, *when melted in vacuo*, will give magnetic properties approaching very closely to those obtainable with electrolytic iron.[22] At the present time there are obtainable a number of commercial grades of iron that come within these specifications (see Figs. 21 and 22).

The author fully realizes the difficulties that have to be overcome before the vacuum iron can be used in the industries in competition with materials available at the present time. Suggestions have, nevertheless, already come from several sources for its employment in places where its high cost of production is not of vital importance, and there is no doubt but that its usefulness in limited fields will increase as time goes on. However, the author would like to suggest that the results here given be

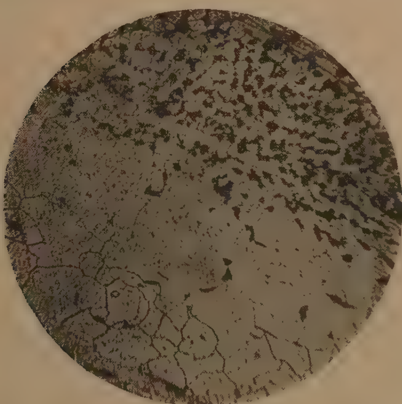

FIG. 21.—SWEDISH CHARCOAL IRON AS CUT FROM PLATE ANNEALED AT 945°C. CARBON CONTENT 0.163 PER CENT. MAGNIFICATION × 40 DIAMETERS.

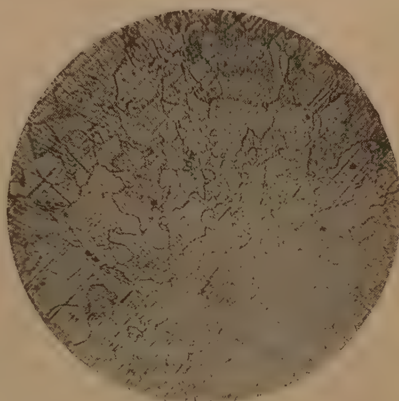

FIG. 22.—SWEDISH CHARCOAL IRON, AFTER BEING REMELTED IN VACUO. AS FORGED. CARBON CONTENT 0.008 PER CENT. MAGNIFICATION × 40 DIAMETERS.

partly considered as a new indication of the possibilities obtainable in the realm of magnetic properties, rather than as the final word regarding certain properties of iron-silicon alloys that can be turned into commercial use at the present time.

In conclusion the author wishes to acknowledge his indebtedness to many persons in various departments of the University of Illinois who have given their time and thought to the investigation. In particular, he wishes to mention Prof. E. B. Paine, acting head of the Electrical Engineering Department; W. A. Gatward, Fellow in the Engineering Experiment Station, for his conscientious work in connection with the magnetic testing, and Messrs. J. M. Lindgren and F. H. Whittum of the Chemistry Department for the chemical analyses.

[22] *Bulletin No. 72, Engineering Experiment Station, Illinois University*, pp. 33, 44 (1913–14).

Discussion

John A. Mathews, Syracuse, N. Y.—Concerning the critical composition of the steel containing 2.50 per cent. of silicon, I may say that 12 or 13 years ago we made some of these alloys. They were not as pure as the alloys made by Mr. Yensen, nor were they melted *in vacuo*, but one of them contained exactly 2.50 per cent. silicon, and about 0.25 per cent. carbon. It gave us no trouble whatever in hammering to a billet or rolling into bars, nor even when rolling into ¼-in. wire rods. The steel in question was practically free from everything except carbon or silicon and contained no manganese sulphur or phosphorus worth mention. The alloy containing 2.50 per cent. of silicon worked perfectly well, as well as one containing 0.5 per cent. silicon, and it seems to me that further work should be done by Mr. Yensen on the physical properties of alloys at this seemingly critical change point.

W. E. Ruder, Schenectady, N. Y.—We have made silicon steels of practically every composition and have never obtained any material under 5 or 6 per cent. of silicon which was brittle. This may be due, of course, to the presence of carbon, in some of the commercial materials, and I presume that Mr. Yensen would argue that the presence of this carbon might prevent inter-granular brittleness. However, all of our silicon steels, up to 5 per cent. of silicon, by treatment in hydrogen at temperatures near the melting point, can be made to show inter-granular brittleness. Furthermore, the material made brittle in this way could be restored by treating at the same temperature in hydrogen containing a hydrocarbon vapor. I discussed this matter some years ago in a paper on inter-granular cement. It therefore appears likely to me that Mr. Yensen's material is probably brittle because of some chemical component unknown to him, rather than because of a newly discovered critical point.

J. W. Richards, So. Bethlehem, Pa.—I call attention to Fig. 4 in which a very quick drop of mechanical properties is shown above 2.50 per cent. of silicon. This may indicate a very steep drop in qualities between 2.50 and 2.56 per cent., as mentioned by the author.

As to the formula $Fe_{19}Si$, I think we ought to be cautioned against too quickly applying formulæ to metallic alloys. There are no alloys for which a formula cannot be worked out, but I think there are very few for which a formula means anything.

T. D. Yensen, Urbana, Ill. (communication to the Secretary*).—In reply to the remarks made by Dr. John A. Mathews, that alloys made by him containing exactly 2.50 per cent. silicon forged perfectly, I wish to say that his results are in perfect accord with ours. By referring to

* Received Mar. 6, 1916.

Fig. 12 it will be seen that our alloys are quite normal even beyond the 2.50 per cent. mark. It is only when the silicon content becomes 2.55 to 2.60 that the brittleness appears, and even in this range several alloys were obtained that forged perfectly. The results of the mechanical tests, however, showed that they were all comparatively brittle. This same statement also applies to the 5.0 per cent. alloy. While somewhat harder than the lower alloys, it forged without difficulty.

I agree with Professor Richards that a chemical formula should not be applied to alloys without definitely ascertaining whether it actually does apply. It was for this reason that I made the statement in my paper that I did. I merely called attention to the fact that the composition of the alloy at the critical point corresponds to the formula $Fe_{19}Si$. Because critical points in alloys are commonly associated with the formation of chemical compounds, I suggested the possibility that some compounds, not of necessity $Fe_{19}Si$, might have formed at the critical point in question, and caused the peculiar behavior of the alloys in this region.

Mr. Ruder said that the brittleness referred to in the region of 2.55 to 2.60 per cent. silicon might be due to some unknown chemical component. If this explanation is correct, it seems strange that we should find this brittleness within these narrow limits only. The alloys were all melted under the same conditions, under a pressure of 0.5 to 1.0 of Hg, the gas remaining in the furnace being CO. The subsequent treatment was also the same in all cases, the annealing being done *in vacuo*, in the absence of any hydrogen-carbon gas. Therefore, it does not seem probable that we should have obtained the peculiar behavior referred to in this restricted range only, in a large number of cases, at different times, unless this behavior is caused by the particular proportion of iron and silicon.

With regard to the density of the vacuum alloys, Mr. Ruder said that the gain in magnetic properties obtained by means of the vacuum alloys is somewhat offset by the increase in density of the latter. I have measured the density of a commercial steel, containing about 3.50 per cent. silicon, and also a silicon-vacuum iron, containing about 3.0 per cent. silicon, and found the latter to have a density 2 per cent. higher than the commercial steel. With the same silicon content the difference would probably be in the neighborhood of 1.5 per cent. Furthermore, Mr. Ruder evidently overlooked the fact that my hysteresis results are given in energy loss per unit volume, and not in energy loss per unit weight. Consequently when compared with the commercial silicon steel on the basis of energy loss per unit weight, the results will be even higher in favor of the vacuum iron than my results would indicate.

The Pacific Coast Iron Situation

The Iron Ores of California and Possibilities of Smelting

BY CHARLES COLCOCK JONES,* B. S., LOS ANGELES, CAL.

(San Francisco Meeting, September, 1915)

In any discussion of this very large subject we are confronted at the outset with so many obstacles that at best only a fragmentary and rather disconnected presentation can be made of it, and my hope is that building from the few facts I have to offer a more complete knowledge may be secured.

In the whole intermountain and Pacific Coast region, west of Pueblo, Col., and from Canada to Mexico, there is no production of pig iron, and steel production is confined to the manufacture of a small amount of open-hearth steel from scrap. When we consider the area, the resources, and the several centers of population, this condition seems anomalous, even in the face of every counter argument as to cheap water transportation, or other means of transport from the Atlantic Seaboard or from the iron and steel centers of the Middle West.

This comatose condition, if such a term may be applied, is certainly not due to a lack of all the essential materials and factors for success in iron and steel smelting, but is rather, I think, due largely to the general fact that it is only within a comparatively few years that the centers of population on the Pacific Coast have attained a size that would entitle them to the position of distributing or freight centers, or that the "back country" has been enough built up to make it sufficiently interesting to the railroads to foster trade from the local centers to the interior points rather than seek to supply such local centers and the interior by a transcontinental haul.

Ore Supply

Edwin C. Eckel, in his monumental work on *Iron Ores*, has but little to say concerning the Pacific Coast deposits. He mentions (p. 269) those of Minaret, Madera County, and the Eagle Mountains, Riverside County, Cal., "perhaps 50 to 100 million tons each;" the deposits in Shasta County, worked for the electric furnace on Pitt River; and

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* Consulting Engineer.

"ores in Oregon and Washington" which "have been slightly developed;" and while he admits that (by reason of unfavorable conditions, as to labor and fuel, for the manufacture of iron) there has been comparatively little prospecting for iron ores in the Pacific States, he thinks that certain geological types of iron-ore deposits, which have been highly productive in the East, are probably wanting here—for instance, sedimentary beds like the Clinton, and brown-ore deposits like those of the Appalachian region. And in his final summary of American iron-ore resources (p. 349) he cites Hayes' estimate of 126,000,000 tons of available ore "for the Rocky Mountain and Pacific States;" pronounces it too low, since Utah alone contains more; and finally puts down the Western States for a minimum of 300,000,000 and a possible maximum of 700,000,000 tons. His estimate for the whole United States being 5,200,000,000 minimum and 7,500,000,000 maximum, the proportion assigned to the great area between the Rocky Mountains and the Pacific is notably small.

Mr. Eckel does not cite, in his Bibliography of California deposits, *Bulletin* 503, *U. S. Geological Survey*, by E. C. Harder, on Iron Deposits of the Eagle Mountains, published in 1912. In the light of the Harder report on the Eagle Mountains and other data, I think the Eckel estimate of from 300,000,000 to 700,000,000 tons can be safely applied to the Pacific States alone, excluding the tonnages of Utah and the other Rocky Mountain States; and, from our present knowledge, most of it is in Southern California.

At the annual meeting of the American Mining Congress held in Los Angeles in 1910, the writer, in an address on the subject of Iron Ores of the Southwest, called attention to the large available deposits of high-grade iron ore in Southern California and adjacent territory, and previously in the *Engineering and Mining Journal* of April 17, 1909, had described technically the iron deposits in the Providence Mountains of San Bernardino County, California. The U. S. Geological Survey published in 1908, *Bulletin* 338 on The Iron Ores of the Iron Springs District, Southern Utah," and in 1912, *Bulletin* 503, Iron Deposits of the Eagle Mountains, California. Descriptions of various other iron deposits are scattered through technical publications.

Epitomizing the subject in 1910, I came to the conclusion that there are 200,000,000 tons of available, high-grade iron ore and double that amount of probable ore in southern California. In addition to this an equal amount can be counted on from Lower California and Mexico, which will naturally seek an outlet to the northward.

This estimate does not include the Utah deposits.

In 1908 and 1909, the late Edward H. Harriman acquired the Eagle Mountain iron deposits, in Riverside County, 50 miles north of Mecca, a station on the Southern Pacific Railroad, 140 miles east of Los Angeles,

with the publicly avowed intention of establishing a steel works at Los Angeles. The U. S. Geological Survey states as a "moderate estimate" that there are 60,000,000 tons of iron ore in this deposit. Later, what is known as the Cave Canyon iron deposit, 200 miles east of Los Angeles on the Salt Lake Railway, containing 20,000,000 tons, was bought by the same interests. The untimely death of Mr. Harriman in 1909 put a damper on further development, and his vision is yet to be realized.

Near Kelso, 236 miles east of Los Angeles, is the Vulcan iron deposit, containing from 6,000,000 to 10,000,000 tons of high-grade, soft hematite ore of Bessemer grade.

In the Kingston Mountains at the California-Nevada line, north of the Salt Lake Railway, are other large deposits.

The Colorado Fuel & Iron Co. owns a deposit of 12,000,000 tons 10 miles west of Silver Lake on the Tonopah and Tidewater Railway.

There are large amounts on the Santa Fe Railway in San Bernardino County south of Newberry, north of Amboy and south of Blythe Junction, and numerous other deposits are known in the desert region. The three most practically available deposits, viz.: the Eagle Mountain, Cave Canyon, and Providence Mountain deposits, represent an aggregate tonnage of 125,000,000 tons, and have the characteristics necessary to the best possible furnace mixture for economical smelting. These ores run from 4 to 10 per cent. higher than standard Lake Superior iron ores. The bulk of the tonnage is within the Bessemer limit in phosphorus and, lacking moisture, there is no cost of drying before transportation, or freight cost on 10 per cent. of moisture as with Lake ores. The average analysis of Providence Mountain iron ore is:

Iron, 64.82; phosphorus, 0.044; silica, 3.04; manganese, 0.278; alumina, 0.568; magnesia, 0.201; lime, 0.444; sulphur, 0.059; titanium, none; volatile, 1.85 per cent.

Consumption

According to one of the leading buyers of iron and steel, the following is a conservative estimate of the present approximate yearly consumption on the Pacific Coast:

	Tons
Pig iron	100,000
Reinforcing steel for concrete buildings	30,000
Sheets	136,000
Plates	217,000
Corrugated	15,000
Bar	175,000
Structural	250,000
	923,000

Probably the aggregate is more nearly double this estimate. Complete figures would include the large amounts of cast-iron water pipe, oil-well pipe, railroad steel and other steel products not listed above.

Costs of Production

At some future time there will certainly be one iron-smelting center in Southern California; and there may possibly be a second at San Francisco and a third at Seattle, each having its own distinct province from which to draw its supplies of raw materials; but at the present time Southern California seems to have the advantage in the combination of ore, fuel, labor and climatic conditions for economical production.

The statement that conditions justify the establishment of at least one plant, based on the consumption of iron and steel products on the Pacific Coast will be borne out, if it can be shown that these products can be manufactured locally at a price to compete with the supply from other smelting centers.

In the following discussion some of the figures relating to Pittsburgh are used as given in a recent publication by the Merchants' Association of New York on the iron and steel industry, supplemented by personal knowledge, and figures from *Mineral Resources of the United States*, 1913, published by the U. S. Geological Survey, and also figures from the report of the Commissioner of Corporations on the Steel Industry, 1911, and costs of making pig iron from testimony before the Ways and Means Committee of the House of Representatives in 1906.

Pittsburgh is the center of the iron and steel industry in the United States and any costs on the Pacific Coast that can be shown to approximate Pittsburgh costs will demonstrate not only the feasibility of the industry, but the certainty of profits.

The difference in mining and transportation costs between Lake Superior ores and California ores is due to a number of reasons, principal among which are the following: The Lake Superior ores are obtained either by expensive underground mining or by open-cut work in which large amounts of overburden have to be removed at considerable cost; and, on account of climatic conditions, the shipping season is limited to seven months of the year. Transportation is a combination of rail, water and rail, entailing several handlings of the ore.

In the case of California ores, we have large surface exposures of orebodies, capable of being quarried and loaded on cars by steam shovel, and handled thence direct to the furnaces. The mining cost of California iron ores will be on the same low basis as that of the porphyry copper ores, or from 20 to 30c. per ton. The figure of 85c. in the following estimate is ample to cover both royalty, or sinking fund, and mining. Climatic conditions are such that work can be carried on every day in

the year with the exception possibly of a few stormy days; and California ores have the added advantage of a higher content in iron and a lower amount of gangue to be fluxed, resulting in greater efficiency, or tonnage product of a furnace and decreased cost of flux, per ton of iron produced, all of which factors materially lower the operating cost per ton of pig iron produced.

TABLE I.—*Comparison of Costs of Iron Ore and Transportation at Pittsburgh and Los Angeles*

	Pittsburgh Lake Superior Ores	Los Angeles California Ores. Assuming Max. Haul 260 Miles at 0.005c.
Rail and Lake rate to Lower Lake Ports	$1.15	....
Rail freight rate, Lower Lake Ports to Furnace location	$0.88	....
Rail freight rate, Eagle Mountains, Providence Mountains, and San Bernardino County ores	....	$1.30
Total transportation, costs per ton	$2.03	$1.30
Average mining cost per ton	$2.40	$0.85
Total transportation and mining cost per ton	$4.43	$2.15
Percentage iron yield per ton ore, non-Bessemer grade, per cent	51.50	57.00
Percentage iron yield per ton ore, Bessemer grade, per cent	55.00	64.00
Total transportation and mining cost per ton pig iron, 95 units iron, non-Bessemer grade	$8.17	$3.56
Total transportation and mining cost per ton pig iron, Bessemer grade	$8.05	$3.19

Comparing the Bessemer ores alone, as the bulk of the California ore is of that quality, the advantage of Los Angeles over Pittsburgh, in cost of iron ore per ton of pig iron is $4.86.

The average mining cost for Lake Superior ores is taken from *Mineral Resources of the United States*, 1913, U. S. Geological Survey, and is exclusive of any transportation costs.

Coke Supply

Of equal importance to the iron ore is a supply of suitable coal for coking. There are probably good coking coals in Alaska and Washington that could be placed in the Los Angeles market at a reasonable price; and there are large coal fields in southern Utah at present without railway connection, undoubtedly containing coking coals; but at the present time the coking coal of Carbon County, Utah, is the most available by reason of rail connection. The Carbon County, Utah, coal, at Sunnyside in the Book Cliffs coal field, compares favorably in coking and

chemical qualities with some of the Eastern coals and a comparison of costs on that basis will be made.

An average analysis of Sunnyside coal and coke is as follows:

	Coal, Per Cent.	Coke, Per Cent.
Moisture	1.27	0.13
Volatile matter	39.13	0.83
Fixed carbon	54.62	88.96
Ash	4.98	10.07
	100.00	99.99

The metallurgical coke for the Utah and Montana copper smelters is supplied from the Sunnyside field and its good qualities are thoroughly proven. A description of the Book Cliffs Coal Field was given by Robert S. Lewis.[1]

In the report of the Commissioner of Corporations on Steel Industry, 1911, the cost of coke at Pittsburgh has been shown to be $2.97 per ton of pig iron, and from that cost it is estimated that the cost of byproduct coke, provided a profitable market for the byproducts could be secured at Pittsburgh, would be $1.94 per ton of pig iron. The above figure of $2.97 is borne out by the current quotations on Connellsville coke, May 1, 1915. As compared with the cost of byproduct coke at Pittsburgh, the following costs are conservative for Los Angeles, where a ready market could be found for the byproducts and where a number of chemical industries could follow the establishment of a byproduct coke plant, utilizing the coal tar, ammonia and gas.

Table II.—*Cost of Coke at Los Angeles or Southern California Points*

Mining cost 1.65 tons Utah coal at 90c. per ton		$1.48
Transportation 1.65 tons at $4.25, 850 miles at 0.5c		7.01
Total cost coking and byproduct operation per ton coke		1.40
		9.89
Credit from sale of byproducts per ton coke, 8,000 cu. ft. gas at 15c	$1.20	
14 gal. tar at 4c	0.56	
24 lb. sulphate ammonia at 3½c	0.84	2.60
Net cost of coke per ton		$7.29
Total cost of coke per ton of pig iron (1.10 tons per ton pig)		$8.02
Cost byproduct coke Pittsburgh, minimum		1.94
Maximum advantage of Pittsburgh in cost of coke per ton of pig iron		6.08
Total cost of coke per ton of pig iron (1.10 tons per ton pig)	8.02	
Or at average cost of coke	2.97	
Average advantage of Pittsburgh in cost of coke per ton of pig iron	$5.05	

[1] *Trans.*, l, 658 to 678 (1914).

The building of the Alaska Railroad into the Matanuska coal field by the government will open that coal to Southern California; and, if the claims regarding its coking qualities are substantiated, it will afford a supply of coking coal cheaper than from any other section.

In Table II the coking question has been treated from the standpoint of making byproduct coke at tidewater in order to improve the coke, secure the byproducts, and have a source of supply of power in the waste gases for use in a steel plant independent of any one coal field; but the coke supply can also be considered from the standpoint of beehive-oven coke made in the coal fields and transported to tidewater, if the cost of such coke can be shown to be less than that of byproduct coke.

I should observe that my estimates are made on a rational freight rate of 0.5c. a ton a mile. The present published rates on coal and coke from the interior are higher than that, being still based on "all the traffic will bear;" and, since the amount handled is small, no attempt has been made to secure a lower rate. That 0.5c. a ton a mile is a reasonable maximum is shown by the fact that that rate is in existence for phosphate rock and other material, of a like or higher value than coal, from this intermountain region to tidewater. I mention this matter because it is one of the "scarecrow arguments" that have to be met. When the traffic justifies it, the rate can be secured, through the usual channels.

Limestone Supply

Available to every railroad entering Los Angeles, at distances of from 20 to 150 miles, are ample deposits of limestone suitable for blast-furnace flux. This rock can be quarried and brought to tidewater at as low a price as that used in Pittsburgh, and from many of the deposits for a less cost. Even at a greater cost per ton of limestone, the cost per ton of pig iron would still be as low as the Pittsburgh cost on account of the greater purity of the iron ores and consequent smaller amount of lime used for flux. In my estimate of final costs, the Pittsburgh figure will be applied as fully covering the limestone cost.

Application of Above Figures and Comparison of Cost of Pig Iron at Pittsburgh with the Estimated Cost at Los Angeles

In 1906, before the Ways and Means Committee of the House of Representatives, Judge Gary testified that for all the furnaces of the United States Steel Corporation the total cost of making a ton of pig iron, based on the market prices of iron ores, coke, limestone, cinder and scale, and including maintenance and depreciation of furnaces, was $15.30 per ton of pig iron, divided as follows:

TABLE III.—*Cost of Pig Iron at Pittsburgh*

Iron ore per ton $4.70, per ton pig iron	$8.62
Coke per ton $3.93, per ton pig iron	4.15
Limestone $1.06, per ton pig iron	0.49
Scrap, cinder and scale	0.27
Labor and maintenance	1.37
Depreciation of furnaces	0.40
	$15.30

From these figures he said the following deductions should be made for net profits:

On ores in pig iron	$2.04	
On coke in pig iron	0.60	
On transportation	1.07	$3.71
Actual cost of pig iron		$11.59

Assuming other costs than the ore, coke and flux to be the same at Los Angeles and Pittsburgh, we have the following as a conservative statement of the estimated cost of a ton of pig iron at Los Angeles, or tidewater.

TABLE IV.—*Estimated Cost of Pig Iron at Los Angeles*

From Table I—Total transportation and mining cost per ton pig iron, Bessemer grade ore	$3.19
From Table II—Total cost of coke per ton pig iron	8.02
Limestone	0.49
Scrap, cinder and scale	0.27
Labor and maintenance at furnaces	1.37
Depreciation of furnaces	0.40
	$13.74
In *Mineral Resources*, 1913, Part 1, p. 321, the government gives as the average price per ton of pig iron at the Pennsylvania blast furnaces for 1913	$15.36
And for Alabama furnaces	12.08

Accepting for the purposes of this discussion the figure given by Mr. Gary of $11.59 as the net cost of pig iron for all the furnaces of the United States Steel Corporation in the year 1906, and deducting it from the estimated cost at Los Angeles of $13.74, we have a difference of $2.15 per ton in favor of Pittsburgh, but the railroad freight rate on pig iron from Pittsburgh makes the net cost of pig iron at least $20.95 f.o.b. Los Angeles and selling price $23. to $25. per ton.

In regard to freight rates on pig iron and steel products to Pacific Coast points, the published rate from Chicago and Birmingham is $9 per ton, and there is no through rate from the Pittsburgh district, and from Pittsburgh to Chicago takes a local rate of $3.15, making the total freight from that point $12.15 per ton to California terminal points.

Steel products have a uniform freight rate of $16 per ton, Pittsburgh to Coast points. In my calculations an average freight rate of $9 has been used, being the rate from Chicago or Birmingham to Coast points.

It should be noted particularly that our comparison of costs is with the costs of a corporation owning mines, railroads, vessels and smelting plants and its costs are necessarily lower than the merchant-furnaces in Pennsylvania which buy ore and coke, and a deduction from Tables I and II more nearly states the relative costs.

From Table II, advantage of Pittsburgh in cost of coke, per tonof pig iron.....	$6.08
From Table I, advantage of Los Angeles in cost of ore, per ton of pig iron......	$4.86
Net advantage of Pittsburgh in cost of ore and coke per ton of pig iron........	$1.22

If, however, in all fairness to Southern California, the figure $5.05, the *average advantage,* instead of $6.08, the *maximum advantage* of Pittsburgh in cost of coke per ton of pig iron is used, as shown in Table II, we have 19c. only as the *net advantage* of Pittsburgh in cost of ore and coke per ton of pig iron. *In other words, iron ore is the costly item and coke the cheap item at Pittsburgh, while in California the costly item is coke, and iron ore is the cheap item, with the difference in the cost of the two materials about equalized.*

From Table I, the cost per unit of iron, Lake Superior iron ore, Bessemer grade at Pittsburgh, is 8c.; the cost of California iron ore, per unit of iron, at Los Angeles is 3.33c.

Markets and the Cost of Reaching Them

At the present time the entire Pacific Coast is dependent upon the East and South for its supply of pig iron and steel, with the exception of a small amount of open-hearth steel produced from scrap at San Francisco and Los Angeles. The imports of foreign iron and steel are a negligible quantity and are never likely to be of any importance on account of the cost of transportation.

Therefore, as our principal market, we have this Pacific Coast province, comprising the States of California, Oregon and Washington, which can only be reached by the eastern manufacturer after paying a minimum transcontinental freight of $9 per ton, or a minimum water freight from Atlantic ports of $6 per ton.

Los Angeles is the great lumber market of the Pacific Coast, and most of the vessels in the trade are without cargo on the return trip and would welcome the opportunity to carry iron and steel products to San Francisco, Portland, Seattle and Vancouver at low water-rates. The markets of South America are open on more than equal terms; for tonnage through the Panama Canal from the Atlantic seaboard to the west coast of Mexico and Central and South America must pay the toll of $1.20 per ton.

Simply stated, pig iron can be made in the regular blast furnace way at Los Angeles for less than $14. No Pennsylvania pig iron, either by transcontinental freight or by rail to New York and via Panama Canal, can be put into this market under $19 per ton. Alabama pig might be put in at a slightly less cost, but its quality would not place it in the same class as pig iron made from California ores.

Geology

The principal Southern California iron deposits are of the contact-replacement type formed at or near the contact of an intrusive igneous mass with sedimentary strata, usually limestones. Their occurrence has been fully discussed in Iron Ores of the Iron Springs District, Southern Utah, by C. K. Leith and E. C. Harder, *Bulletin* 338, *U. S. Geological Survey*, 1908; Iron Ore Deposits of the Eagle Mountains, California, by E. C. Harder, *Bulletin* 503, *U. S. Geological Survey* (1912); and An Iron-ore Deposit in the California Desert Region, by C. C. Jones, *Engineering and Mining Journal* (April 17, 1909); and to these accounts I will add only the following note, bearing somewhat on a recent discussion in the Institute.

The Providence Mountains of Eastern San Bernardino County present the greatest uplift and cross-section of strata found between the Grand Canyon of the Colorado in Northern Arizona and the Sierra Madre range in Western San Bernardino County, and afford an opportunity to study a section approximately 4,000 ft. thick, from the underlying granitic and schistose rocks up through a sedimentary series predominantly composed of limestones with subordinate beds of shale and quartzite. This has been brought to the attention of the U. S. Geological Survey; and I believe an extended study of the region will fill several gaps in our knowledge of the geology of this part of the desert region.

The general strike of the sedimentary beds is northeast-southwest with a slight dip to the east. The central core of the mountain, exposed in canyon from base to top, and forming the highest peaks, is a core or plug of rhyolite, a mile or more in diameter. At the southwest end of the mountain the sedimentary beds, lying horizontal or dipping slightly to the east, and retaining this position undisturbed, have been cut by a large quartz-monzonite mass, the contact between the monzonite and the limestones being practically vertical. I have indicated on the accompanying sketch map the general conditions.

The specific fact I wish to bring out for the attention of those interested in the discussion on The Origin of the Garnet Zones and Associated Ore Deposits,[2] by Waldemar Lindgren, and Recrystallization of Lime-

[2] *Trans.*, xlviii, 201 to 208 (1914).

stone at Igneous Contacts,[3] by C. K. Leith, is the different alterations sustained by the same series of limestones in adjacent areas in contact with the same igneous intrusive rock mass.

In the sketch map, I have indicated conditions 9 miles southeast of Kelso. *R* is the central core of this part of the Providence Mountains, composed of rhyolite, and detached small rhyolite exposures are shown at several places in the limestones marked *R*. *XY* is the line of contact between the sedimentary series and the quartz-monzonite mass.

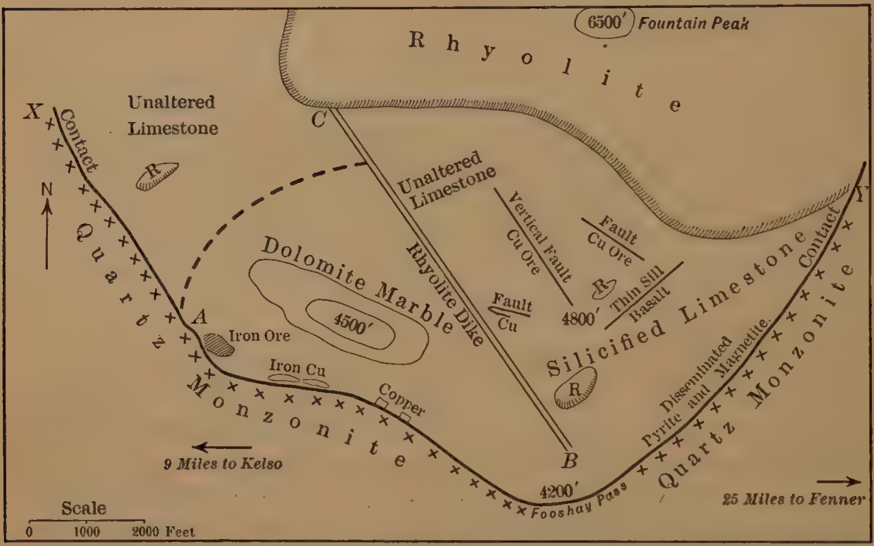

Fig. 1.—Sketch Map Showing Geological Conditions in Eastern San Bernardino County, Cal.

The contact of the monzonite, cutting sharply the practically horizontal beds of limestone, runs for a distance of about 4 miles. *BC* is a vertical dike of rhyolite porphyry, 100 ft. thick, an offshoot from the central core. The point marked *A* seems to have been the center of activity, resulting in a replacement of the limestone by iron, over an area of four acres, with smaller orebodies along the contact to the east and with a great dissemination of grains and irregular replacements of iron ore in the limestones. The original bedding planes of the limestone are still preserved in the beds of iron ore; and in the area *ABC* the limestones have largely lost their bedded structure and a mass of crystalline dolomite or dolomitic marble has resulted. A fuller description of the geologic features is given in my article in the *Engineering and Mining Journal* cited above. The end of the dike *BC* is not seen in actual

[3] *Trans.*, xlviii, 209 to 215 (1914).

contact with the monzonite at *B*. To the west of dike *BC* is an area of dolomite, as indicated, of the following analysis:

	Per Cent.
Insoluble siliceous	0.43
Aluminum and iron oxides	0.24
Calcium carbonate	63.08
Magnesium carbonate	35.97
	99.72

To the east of dike *BC* is an area of limestones between the rhyolite and monzonite, a portion of which is seemingly unaltered, especially just east of point *C*, where, in contrast with the dolomite west of the dike, the blue unaltered limestones preserve perfectly their bedded structure, and the larger portion of the area, especially at and for several hundred feet north of the contact, is largely silicified. An analysis of a typical sample shows:

	Per Cent.
Silica	48.0
Lime	25.8
Iron oxide	4.3
Alumina	12.9
Carbonic acid	5.0
Undetermined	4.0
	100.0

Much analytical work is needed to determine the composition of the original unaltered limestone. While I am not prepared to discuss thoroughly the phenomena involved, my impression is that the original limestones were calcitic, and that ascending heated waters, following the intrusion of the monzonite, decomposed its ferro-magnesian members, deposited the iron as a replacement of the limestone near the contact, and dolomitized the limestones between the contact and the dike *BC*. But the reason of the different action east of the dike *BC* I leave to those better qualified to determine. In the east area are a number of vertical faults, some of great length and some short, nearly all of them showing a slight mineralization, chiefly of copper minerals in calcite veins, the gold and silver contents being negligible. Unfortunately the work done on these veins has not been productive; otherwise a fuller study would, perhaps, have been made of the conditions I have indicated above. The only working mine in the immediate region is situated 9 miles northeast of Fooshay Pass on the same contact, modified by some pegmatite dikes, at what was formerly known as the Bonanza King mine, where several million dollars worth of silver ores were extracted from irregular bodies in the limestones.

Discussion

D. A. Lyon, Salt Lake City, Utah.—For some years I have been interested in the iron-ore deposits of the Pacific Coast, and I feel very much indebted to Mr. Jones for the information he has given us on this subject. I think that we are also very much indebted to him for having presented this paper, as he has given us reliable data in shape of facts and figures, which is just what we need in considering whether or not it would be commercially feasible to attempt to establish an iron and steel industry on the Pacific Coast. Personally, due to lack of market, I do not think it is a commercially feasible proposition at the present time. Mr. Jones states that pig iron can be produced on this coast as cheaply as it can be produced in Pittsburgh. Even so, what can be done with it after it is produced? To begin with, iron produced on the Pacific Coast has to compete with iron produced in China. Last year I had occasion to investigate the subject of the market for iron and steel on the Pacific slope, and found that it was possible for the producers of Chinese iron to sell it for much less than the price for which iron can be produced in the West, and also for less than eastern pig iron is now sold for, even when shipped through the Panama Canal. I was informed that the agents for Chinese iron had instructions to sell it for $1 or $2 less per ton than that asked for other irons on the market. Of course, I realize that the competition of Chinese iron may not be important, but even if it is not, and even though pig iron can be produced at the cost given by Mr. Jones in his paper, I do not see any immediate chance of its being so produced, due to lack of market.

C. C. Jones, Los Angeles, Cal.—Mr. Lyon is right. It is a fact that Chinese pig is being sold in San Francisco on some large contracts at $18.50 a ton, while $19 is the usual price. We ought to produce iron on this coast at $14. Personally I do not think Chinese iron is ever going to be an important factor. It is a bugbear out here now. You are not going to have much Chinese pig iron in this country. Our little friend Japan is going to absorb everything it can. The price of Chinese pig iron at $18 is about as low as it will ever get. If we can show you that we can produce this iron here at the same price as at Pittsburgh, I think you will have a good opportunity to compete with anything China or India can ever do. Costs there are very low now. They are going to come up by leaps and bounds in the next few years. The labor cost over there may be very low now, but I have seen some men from there quite recently who say the costs are going up. Those people can not live under the conditions that are imposed upon them at the present time.

The best market is not in rails and heavy structural steel but in the smaller diversified industries. That is the point I am trying to make. The idea is not to try to take away the steel rail business or the big steel

plate business from Pittsburgh but to install a blast furnace and equipment to make special steel.

Carl H. Booth, Chicago, Ill.—Outside of a lack of a market I feel that there is one other weakness in the situation at this time, and that is the lack of men who know how to do the work. In my rather brief investigation, I find that there are few men on the Pacific Coast really experienced in steel work. With the exception of charcoal and coke, the natural resources are plentiful and the materials required for the manufacture of iron and steel can be obtained at low cost. My chief criticism has been this lack of men with proper experience. I have just come from a trip to Los Angeles. There are a number of machinery manufacturers there who freely complained about the quality of the steel castings produced on the Coast. Most of them are buying their best castings in the East simply because they cannot depend on the quality of the local product. There is no question in my mind that if experienced steel foundry men are induced to come out here, the iron and steel business can be started properly and successfully. It stands to reason that if the same brains and intelligent handling of the proposition were given to it here as in the East, there is no reason why the resources here should not be developed, and there is no excuse for the Pacific Coast buying its castings back East, when the same application and knowledge could be applied here and castings produced that would be just as good. I am talking now about the manufacture of the product after the steel is made, but there has seemed to be, from the investigation I have been able to make in the last three or four weeks, a lamentable absence of experienced handling. I believe that there is already a large market on this coast for steel castings. If you have both the natural resources and the market here, you must also have the ability to take hold of it and deliver the proper product.

Bradley Stoughton, New York, N. Y.—It would seem to me that the discussion of this paper was departing from the established principles of political economy. Mr. Jones has given us in his paper some very carefully culled figures and the question should be, not whether somebody else could supply the material listed by Mr. Jones at a price less than is now being paid, but whether the figures themselves are correct.

The question as to whether China can or cannot send iron to this country at a lower price than is now being paid for it, and the question as to whether the iron foundries of California do or do not employ sufficiently skilled men to make good castings, could be discussed from many different viewpoints, but if the figures presented by Mr. Jones are correct, and if, as stated by him, the greater amount of iron and steel used on the Pacific Coast is now obtained from the East, then there can be no question that there is a real market here.

Looking at the matter from another standpoint: It would appear that the Pacific Coast with the greatly enlarged scope for industrial activity that is given by the operation of the Panama Canal, including manufacturing of many kinds, exporting, and shipbuilding, must use in the future greatly increased amounts of iron and steel products.

D. A. Lyon.—I think Mr. Stoughton has laid too much emphasis on what I said in regard to the competition of Chinese iron. Although we do have that competition at present, it may not, as pointed out by Mr. Jones, affect the pig-iron and steel situation in the future here on the Pacific Coast. We will assume that it will not; but even so, and even if iron can at present be produced as cheaply at certain points here in the West as it can be obtained from Pittsburgh, now that the Panama Canal is open, will we not be able to get iron and steel from Pittsburgh cheaper than we do now?

C. C. Jones.—You would have to pay the freight rate from Pittsburgh to New York, then you would have a freight rate of $4.50 or $6 from New York to San Francisco or Los Angeles. When you add that it makes very little difference.

D. A. Lyon.—Even if we could produce it here as cheaply as they can in Pittsburgh, I still maintain that we do not have the market. We have with us Mr. Lash, one of the most prominent iron and steel men in the country, and I am sure we would be glad to hear from him on this subject.

H. W. Lash, Cleveland, O.—I think, as Mr. Lyon has pointed out, one of the troubles is a market out here. There is a large consumption of iron and steel on the Pacific Coast, but it includes nearly everything that is produced in the line of iron and steel and there is no one plant that could cover all these different grades and shapes without a tremendous outlay. That is one thing that would interfere more than any other with the production of steel in large quantities on this coast.

As for pig iron, I think there is a greater opportunity. There are many foundries in California that have to bring their iron from the East or from England, and there has been some iron sent here from India and China. I think there is a chance to develop the pig-iron industry, though probably not to any very great extent. There is surely quite a consumption here, and a small plant properly located and properly managed ought to be able to compete with the East if the raw materials exist in the quantities represented, and they no doubt do.

J. W. Richards, South Bethlehem, Pa.—I would like to ask Mr. Lash if he will give us his opinion about the steel-casting industry on this coast?

H. W. Lash.—I have had occasion to visit one or two steel-casting plants, one at Pittsburgh, Cal. They have a very good plant and are turning out good product. They say they have all the business they can possibly turn out. Their castings are made from open-hearth steel and while they have not attempted the smaller castings similar to those made in the East, they will no doubt take up that class of work soon. In my opinion the casting business has quite a future here. The matter of delivery is very much in their favor. Customers do not care to wait for the long time of delivery from the East.

C. C. Jones.—I understand that there is an electric steel foundry near Portland which is in operation. It strikes me that the situation here is particularly favorable for the introduction of an electric furnace to make steel castings from scrap, because there is a very large quantity of scrap on hand which can be melted and recast into shapes. I personally believe that right here in San Francisco or Oakland, right on the San Francisco Bay, an electric steel-casting plant could be quite successfully operated.

D. A. Lyon.—I sincerely hope that I will not be considered a pessimist as regards the possibility of producing iron and steel on this coast. In this discussion I am thinking only of large tonnages. What Mr. Jones says is true. There is a chance out here to develop an iron and steel industry which will supply the iron and steel that is needed for local consumption. However, there is not a market at the present time for large quantities. It is true that the sum total of all the various kinds of iron and steel used amounts to a considerable tonnage. A great deal of that, however, is needed to supply parts of machines or implements which are manufactured in other districts of the United States. Then, too, the variety of shapes needed is too great for any one plant to supply. If, therefore, the local demand is to be furnished by local plants, it seems to me that it must be done by small plants. In other words, I do not believe there is a demand at the present time for a large iron or steel plant on the Pacific Coast, such as we find around Pittsburgh.

C. H. Booth.—I believe there are a lot of concerns out here waiting for an opportunity to use steel in place of iron castings. I have made a somewhat detailed study of California conditions. There are a great many companies that would use steel castings if they could get prompt delivery. This is especially true of small steel castings. The heavy castings are made by the plant you spoke of, Mr. Lash, but the smaller castings today are made with converters or crucibles with the exception of one or two electric furnace plants and the quality has not been altogether what could be desired. Outside of delivery and quality, I think other things are mere matters of detail, yet they are important. A great

deal of the heavier iron castings out here will be replaced by lighter castings made of steel, if they can be obtained at a reasonable cost. It looks to me as if the opportunity is a very large one, because you have electric power here at reasonable rates, and steel scrap today in this market costs from $5 to $7 a ton. You cannot buy scrap at any such price in the East.

Mr. Malm.—I notice that Mr. Jones mentions a deposit in Shasta County and I would be glad to have any information regarding what they did up there with their electric furnace.

J. W. Richards.—That company started to make pig iron in an electric furnace and discontinued it, and has recently started up a furnace to make ferromanganese and ferroalloys, and they are now making them very profitably.

Mr. Malm.—They discontinued the manufacture of pig iron up there.

Mr. Randall.—I was told it was due to the high cost of coke. They had some intention of trying to make it in a charcoal furnace and the Forestry Service was trying to sell them some of their wood.

Mr. Willeston.—The reason for their shutting down was due to the fact that they could not get charcoal at a low enough price. They installed a large coke plant, but it proved a failure. They could not produce charcoal economically, and they could not buy it at a reasonable figure, so it put them out of business.

C. H. Booth.—I have just returned from a visit to that plant. I spent a day there about a week ago. I had an opportunity to talk with one of the owners of the plant, and he said he did not believe they would ever attempt to make pig iron again, that it was not commercially practical. They made some very good pig iron but they could not compete with the cost of other pig iron brought in.

C. C. Jones.—These remarks have helped this subject. Mr. Lash is correct in saying the market here is largely one for pig iron. My idea has been that the establishment of a single furnace of 250 tons capacity would lead to so many other things that would not compete with the larger tonnage, that in the end we would have half a dozen little industries that would mean more in wages and in selling profits than tonnage represents. Take the matter of cast-iron pipe alone. In Los Angeles County today there are contracts aggregating $2,000,000 for this cast-iron water pipe. People are going into the business there with small experience, using poor pig iron with a hope of filling some of these contracts. That is Los Angeles County alone, and does not include the contracts for cast-iron pipe that are being let around Stockton, Port-

land, Seattle, and other places. We have a pretty long coast line here, nearly 2,000 miles, that is going to fill up mighty rapidly in the next 20 years. Anything that is based on an increase of population ought to be a success commercially. As Mr. Lyon has said, it is a commercial matter.

Bradley Stoughton.—We have heard it said in the East that the difficulty of establishing an iron and steel industry, or any industry, on the Pacific Coast is one of elasticity of labor supply and I should like to ask if Mr. Jones has considered this question and what conclusion he has reached thereon.

C. C. Jones.—I cannot say that I have looked into the question of the elasticity of the supply, but I have heard it said that Eastern manufacturers would not go into manufacturing here on account of labor conditions. But it is not the supply. Labor is looking for a fight all the time; the union features are very strong here.

Bradley Stoughton.—Mr. Schwab has some very large interests on the coast, has he not?

C. C. Jones.—Yes, the Union Iron Works, for one, and he probably supplies large amounts of structural steel to firms for fabrication.

H. W. Lash.—I have had some little experience out here, during the last two years and have found the quality of labor very good. I believe the average equals that of Ohio and Pennsylvania, and the climatic conditions are surely more favorable. We have neither intense heat in the summer nor extreme cold in the winter. There is no doubt that labor unions are very strong. In some cases they are unreasonable, but I believe the labor situation here is somewhat exaggerated. I also find there is something about California that is rather inviting and that workmen are quite ready to come here if they can be furnished employment similar to that they have in the East. A great many like to come and try it, and when they do come they usually want to stay.

The Thermal Insulation of High-Temperature Equipment

BY P. A. BOECK,* NEW YORK, N. Y.

(San Francisco Meeting, September, 1915)

The thermal insulation of high-temperature equipment for industrial purposes is a subject that has not received from engineers and designers the attention its importance deserves. This may be attributed to a number of causes: In the first place, heat flow is a rather difficult factor to measure under the working conditions of an industrial furnace, and until the comparatively recent introduction into certain operations of the heat balance sheet, with a systematic attempt to account for the discrepancies in the totals, the various causes of heat losses and the reasons for their existence were not brought to light. Moreover, such rapid advancement and radical changes in the design of equipment, especially in metallurgical lines, have occurred that practice has apparently outdistanced the theory of design. In the modern tendency toward greatly increased size of units, effort has seemingly been made to utilize the heat of fuel gases without proper attention having been paid to conserving heat energy and confining it strictly to the points of maximum activity.

The systematic study of the thermal properties of structural materials, which is being carried out by various government bureaus and also by a number of technical societies, is forming the basis for more intelligent and consistent work in the economically important subject of prevention of fuel waste.

Advantages of Insulation

It is not necessary in this practical and utilitarian age to enlarge upon the need of prevention of heat losses, but attention may advantageously be called to the other benefits derived simultaneously with the saving in fuel and the increase in thermal capacity of furnaces when properly insulated.

In most industrial furnaces—which are here spoken of in the broadest sense, indicating general high-temperature equipment for any purpose and heated in any manner—the operations are generally carried on through a temperature range which has been found by practice to give the most ef-

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fective results. If, in preventing the heat being dissipated from the outside of the furnace, the temperature throughout the furnace is made more uniform, the zone of reaction is manifestly greatly increased without additional fuel consumption. Furthermore, overheating at the source of heat, in the effort to bring as much of the furnace as possible into the zone of reaction, is materially reduced by preventing the loss of heat through the furnace walls. This has the ultimate effect of lowering the temperature at the source of heat and greatly increasing the life of the refractories—usually an important factor in high-temperature work. It is evident, too, that in furnaces which are surrounded by metallic casings or other equipment susceptible to the action of high temperatures, the deterioration and damage caused by overheating may readily amount to many times the cost of installing the proper protecting insulation. The humanitarian will also see the advantages gained by the protection of workmen from the unhealthy and unsanitary conditions brought about by being compelled to labor under overheated conditions.

The importance of thermal efficiency of all manner of high-temperature equipment is now being more generally recognized and is receiving much attention.

Heat Flow

Without going into a detailed consideration of the general laws governing the flow of heat, it would be well to call attention briefly to the following facts regarding heat flow, which will be illustrated by reference to the steam boiler as a well-known example:

The U. S. Bureau of Mines has treated[1] this subject in a remarkably clear and concise manner in Bulletin No. 18, to which reference should be had by those desiring to refresh their memories as to the principles of heat radiation, convection, and conduction.

The energy in the form of heat which eventually finds its way into the water in a boiler, for example, is, of course, subsequently available in other forms. That portion, however, which is transferred to the walls of the furnace setting by any of the three methods mentioned, is conducted through the setting, if means are not provided to prevent, and is lost by radiation or convection from the outer surface of the setting. It is the heat lost in this manner and methods for its prevention that will be especially considered in this discussion.

Rate of Heat Flow

The rate of transfer of heat under various conditions, by both conduction and radiation, shows the relative importance of these two methods in influencing heat losses. The amount of heat conducted through a unit

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[1] Kreisinger, Henry, and Ray, Walter T.: The Transmission of Heat into Steam Boilers, *Bulletin No.* 18, *U. S. Bureau of Mines* (1912).

area from one part of the body to another is proportional to the difference in temperature of the two parts; directly proportional to the thermal conductivity of the body through which the heat passes and inversely proportional to the distance between the two parts of the body. In other words, the conduction of heat through a solid body from one plane to another is a direct function of the conductivity of the body and the difference in temperature of the two planes and an indirect function of their distance apart.[2] This is identical with Ohm's law for transfer of electrical energy.

The heat transferred from one body to another by radiation is proportional to the difference of the fourth powers of the absolute temperatures of the two bodies. While this is strictly true only of the ideal "black bodies," the variation is so small that for all practical purposes this relation holds good in ordinary procedure.[3]

Hollow Wall Construction

This relation indicates the reason why, in low-temperature work, such as refrigeration, etc., a hollow wall space is an effective insulator, whereas, in high-temperature operations, the loss of heat by radiation through a hollow wall space is so great that its insulating effect is less than if this wall space were filled with material of rather high thermal conductivity. This has been brought out by Ray and Kreisinger, who again demonstrate that the hollow wall space type of wall construction is much less effective as a means for preventing the loss of heat than a solid wall of any ordinary construction material of equal thickness. This is especially true if the air space in the hollow wall is near the furnace side and becomes highly heated. This is entirely contradictory to the general belief that, since air is a poor conductor of heat, air spaces built into the walls of a furnace will greatly reduce heat loss by radiation. While the heat does travel very slowly through the air by conduction, it leaps over the air space readily, by radiation, because the quantity of heat which passes across the hollow space is a function of the fourth power of the absolute temperatures of the surfaces inclosing it, which loss is enormously increased by rise in temperature.

Furnace Walls

In general, in high-temperature furnace construction, there are two separate and distinct factors which must be considered to produce an effective wall. The first of these is to provide a material having the ability to resist the action of high temperatures, sufficient mechanical

[2] Ray, Walter T., and Kreisinger, Henry: The Flow of Heat through Furnace Walls, *Bulletin No.* 8, *U. S. Bureau of Mines* (1912).

[3] Gurney, Harold P.: Heat Radiation, *Journal of Industrial and Engineering Chemistry*, vol. iii, No. 11, p. 807 (Nov., 1911).

strength, and, possibly, the property of resisting corrosive slags, gases, etc., without spalling or being eroded. Secondly, to prevent the excessive loss of heat due to conduction from the interior of the furnace to the outside, where it is lost. It is rare that a good refractory material is a good insulator; usually it is necessary to augment or back up the refractory with some material having a much lower heat-conducting capacity. As a general rule, light porous substances are good thermal insulators; and, in a great majority of cases, this depends upon their entrapped voids, their apparent density being an excellent criterion of their conducting ability.

Requirements of Insulators

The requirements of the insulating backing for the more refractory lining are rather severe; an ideal insulator would have the following properties: It should be extremely high in insulating value, and sufficiently refractory so that no fusing or excessive shrinkage would take place in that portion which is in direct contact with the highly heated refractory wall. It should not be decomposed or change greatly in volume at that temperature, and it should, furthermore, be of light weight, unaffected by moisture, of convenient form, readily applied by unskilled labor, and low in cost. It should be of such composition as not to react upon or attack either the refractory material or the metal shell of the container, even in the presence of moisture; *i.e.*, it should contain no free acid radicals, and should not be broken or caused to settle by vibration or heat. It should not have high expansion and it should be sufficiently elastic to take up strains between the lining and the shell produced by temperature changes. While this is a rather formidable array of requirements, there are products upon the market which fulfill practically all of them.

Such materials as are being commonly applied to steam-temperature insulation, of which magnesia and asbestos are best known, are, of course, eliminated from consideration in the great majority of cases mentioned here. The lack of more extended practice in high-temperature insulation must be attributed to the fact that effective high-temperature nonconductors have not been generally available.

Amount of Insulation

When the kind of refractory material that is best suited to the furnace has been determined, the next most important item is the thickness of the walls and the nature of the insulating material most suitable, and it is necessary to determine the degree of insulation which will produce the most effective results. Hering has pointed out that, in order to effect a perfect insulation, the furnace should be surrounded with an insulating material that would maintain the same temperature at any point as that

of any adjacent point of the lining; in other words, that there should be a uniform amount of heat loss throughout the lining.[4]

In attempting to obtain perfect insulation, it is entirely possible to over-insulate, causing serious damage to the refractories in the high-temperature zones of the furnace. To maintain the inner walls, it may be necessary to permit a considerable flow of heat through the wall, with a corresponding decrease in the temperature of the refractories at the heated point, in order to prevent their destruction. It would be manifestly impracticable to insulate the roof of an electric steel furnace or open-hearth furnace, for instance, with a heavy layer of insulating material, because the cost of the refractory lining which would be destroyed would be considerably greater than the cost of the heat which would otherwise be lost.[5]

In electric-furnace practice, where extremely high temperatures are encountered and where effective insulation is necessary because of the relatively high cost of current, the question of insulation, or rather over-insulation, has been given more attention than in other lines. F. T. Snyder[6] has presented a detailed study of the nature and amount of insulation which is permissible under various conditions and presents many valuable data which might with advantage be applied to other industries.

The necessity for effective insulation in stoves and other metallurgical equipment has been brought out strongly by Walther Mathesius, before this Institute, in a paper on High Blast Heats in Mesaba Practice.[7]

In discussing the thermal efficiency of the modern hot-blast stove and probable future methods of improvement, A. E. Maccoun, in a recent paper[8] before the American Iron and Steel Institute, points out the advantages of effective insulation and the necessity for thermal insulation to increase the efficiency of the stoves.

The requirements of an ideal insulator have been outlined, and, in order to show the various methods of application which have been worked out (some of which have been in use abroad or in this country for some time), it will be well to select the insulator which apparently most nearly meets the requirements and use this as an example of methods of construction. In the applications shown, attempt was made to cover as wide a field of use as possible, to show the scope of possible applications.

[4] Hering, C.: Thermal Insulation of Furnace Walls, *Metallurgical and Chemical Engineering*, vol. x, No. 2, p. 97 (Feb., 1912).

[5] Lyon, D. A., Keeney, R. M., and Cullen, J. F.: The Electric Furnace in Metallurgical Work, *Bulletin No.* 77, *U. S. Bureau of Mines* (1914).

[6] The Flow of Heat through Furnace Walls, *Transactions of the American Electrochemical Society*, vol. xviii, p. 235 (1910).

[7] *Trans.*, li, 794 to 818 (1915).

[8] Blast Furnace Advancement, *Transactions of the American Iron and Steel Institute* (1915).

Insulation

The insulator used, known as celite, on account of its extremely cellular nature, is a mineral product of a highly siliceous composition and of very light weight, which occurs on the Pacific Coast[9] in an exceptionally pure state. It is composed of numerous hollow cells, and weighs, in its natural rock form, air dried, from 25 to 30 lb. per cubic foot. When this material is ground properly, so as not to destroy its cell structure, Sil-O-Cel powder is produced, which weighs but 8 lb. to the cubic foot and has a thermal insulating power about equal to that of cork, or from 10 to 12 times the insulating power of ordinary firebrick. In other words, a 1-in. layer of this material is the equivalent in insulating value of from 10 to 12 in. of firebrick. Being almost pure silica, its melting point is high, 2,930° F. (1,610° C.), as reported by the Bureau of Standards, and it can be subjected to high temperatures without fear of alteration. It has been found advisable, however, not to use celite as a refractory at extremely high temperatures without some direct protection. This is readily accomplished by using it as a backing material for more refractory and highly conducting bodies. Owing to its remarkable nonconducting properties, the accumulation of heat on its face is so great, owing to the fact that the surface is not cooled by conduction, that a "flash" of flame or gases might easily exceed the melting point of silica and cause failure. If it is protected, however, only modified and uniform temperatures are encountered, which are maintained without risk or damage.

It is possible further to prepare bricks and blocks of various sizes and shapes by sawing the natural material by means of gang saws. Standard 9-in. straight Sil-O-Cel brick made from natural celite weigh from 1½ to 2 lb. each and are equivalent in insulating value to many times their thickness of ordinary firebrick. In crushing strength, these brick withstand over 400 lb. per square inch and are sufficiently strong to stand transportation and handling.

The high insulating value of Sil-O-Cel bricks can be shown by applying the heat of a blast lamp or torch upon their surface for hours, the unheated surface remaining cool enough to permit of handling. Powdered Sil-O-Cel can be tested in the same way by packing it into a shallow box of convenient size.

The cost of these insulating bricks is but little more than that of firebrick, and of the powder about one-third as much, so that the first cost of this insulation is comparatively low. In fact, instances are on record where the entire cost of insulation has been saved in fuel in the first few weeks of operation.

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[9] Deposits worked by the Kieselguhr Co. of America, New York.

Typical Wall Insulation

Because of the variation in form in which Sil-O-Cel products are supplied—that of brick, blocks of various shapes, in powdered form and as a plastic cement—this material is adaptable to almost any form of thermal insulation, as will be shown in the following typical examples:

General Types

In general, there are four forms of construction for high-temperature insulation which can be adapted to almost any character of equipment.

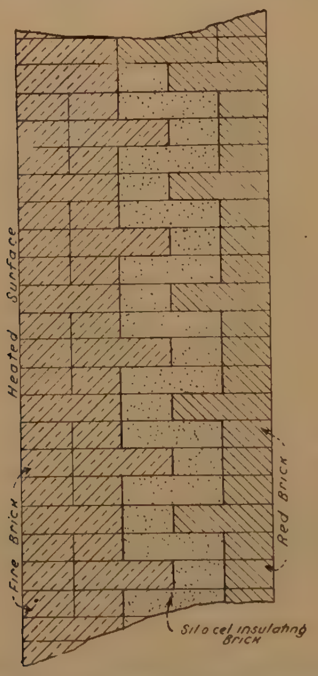

FIG. 1.—TYPICAL WALL CONSTRUCTION, SHOWING INSULATING BRICK LAID IN WALL.

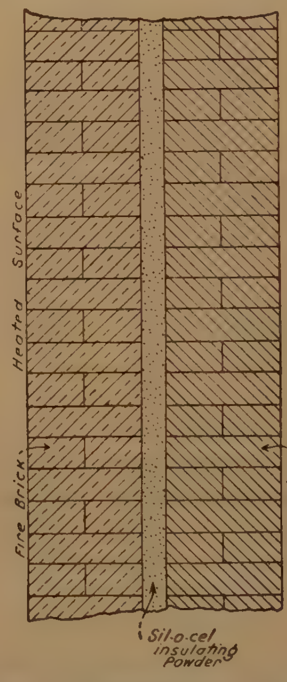

FIG. 2.—TYPICAL WALL CONSTRUCTION, SHOWING INSULATING POWDER IN HOLLOW WALL.

Fig. 1 indicates the usual method of using Sil-O-Cel brick interlaid between a course of firebrick and red brick for the prevention of heat leakage through walls. This form of construction is largely used in boiler settings, bakers' ovens, reverberatory-furnace walls and roofs, etc., and is generally applicable where a strong, solid, nonconducting wall is desired.

Fig. 2 indicates one of the methods of construction of an insulating wall in which an otherwise hollow space is filled with insulating powder. From 2 to 4 in. are usually sufficient. The powder is packed slightly to a density of approximately 12 lb. to the cubic foot, at which point it attains

its maximum insulating value and is not subject to settling or contraction due to either vibration or heat. Where this form of construction has been in severe service in high-temperature furnaces for a period of years no contraction or settling has taken place.

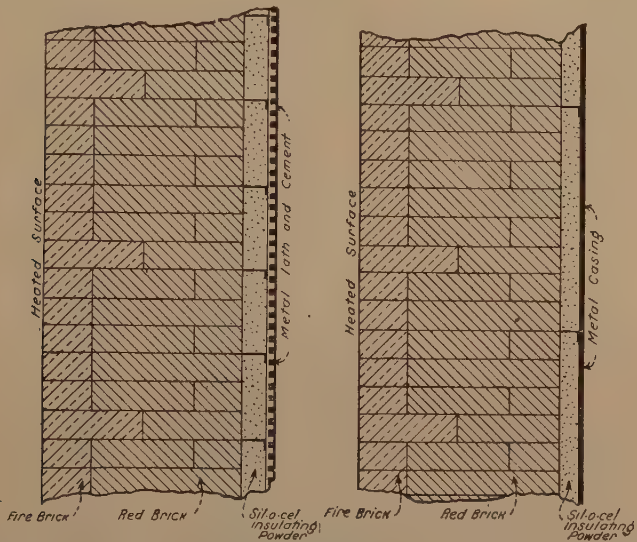

Fig. 3.—Typical Wall Construction, showing Insulating Powder Supported by Metal Lath, Cement Finish, and Powder Supported by Metal Casing.

Fig. 3 indicates the method of insulating brick walls which are already in place. This form of insulation can be applied to old construction as well as new. In this method, expanded metal lath is erected on angle

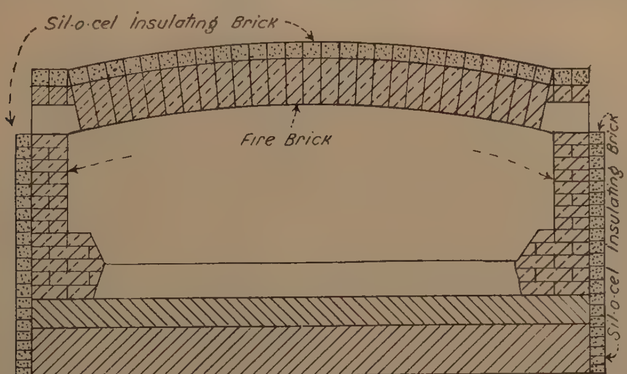

Fig. 4.—Reverberatory Furnace Roof Covered with Insulating Brick.

irons at the required distance from the outer wall and coated on the outside with one or more coats of portland cement plaster, to which a small amount of Sil-O-Cel powder, approximately 20 per cent. by volume,

has been added to give greater plasticity and ease of working and to increase the heat-resisting properties of the cement. Sil-O-Cel powder is packed to a density of 12 lb. per cubic foot between the brick wall and the expanded metal lath. This form of construction is relatively inexpensive and allows of as much insulation as is required, and furthermore gives an absolutely permanent surface of excellent appearance, which can be applied to almost any character of equipment.

A layer of a pulverized material between the brick lining and furnace

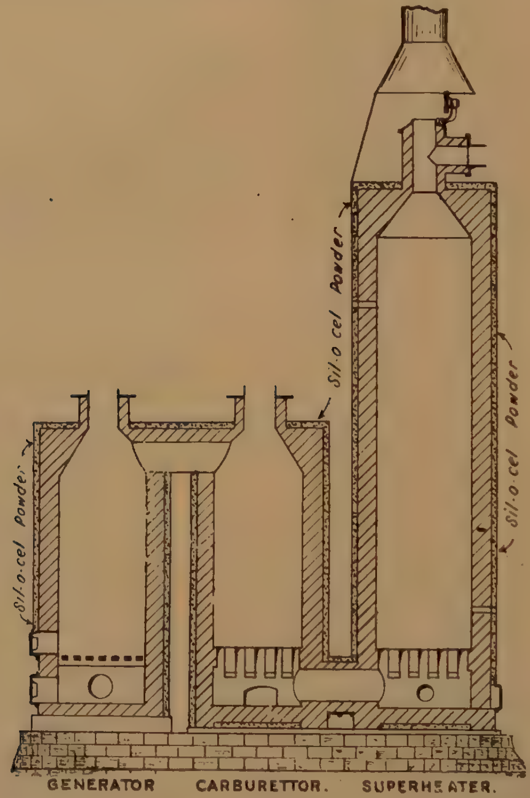

Fig. 5.—Gas-Generator Set Insulated by Means of Powdered Sil-O-Cel.

shell is the form of insulation most generally used in high-temperature metallurgical equipment, such as kilns of all kinds, gas generators, producers, stoves, etc.; in fact, it is universally applicable where the furnace has a suitable shell. In cases where support is required from the shell or setting, a denser variety of Sil-O-Cel, which is not compressible, has been found to be effective. The denser product does not possess the unusually high insulating values of the natural Sil-O-Cel, but offers mechanical advantages for certain construction.

These four forms of construction can be modified to give walls of any

desired thermal efficiency, the respective methods being largely determined by the character of the work under consideration. Specific forms of high-temperature equipment are shown in Figs. 4 to 8.

In reverberatory furnaces Sil-O-Cel has found application as an insulating material for roofs and furnace walls in the manner indicated in Fig. 4.

Fig. 5 indicates the method which is finding universal application for gas-making equipment, showing the powdered Sil-O-Cel tamped between the firebrick lining and the metal shell to a density of about 12 lb. to the cubic foot. This form of construction has been in actual use for a number of years in some of the largest gas plants on the Pacific Coast, in insulating oil-gas generators, superheaters, carburetors and similar equipment. The advantages of Sil-O-Cel in this connection are threefold: In the first place, the heat conducted to the surface of the metal shell which is dissipated by radiation and convection is reduced to a minimum. Secondly, in preventing the access of high temperatures to the steel shell, it prevents oxidation and deterioration of the shell through overheating and gives it the protection which it should have. Thirdly, Sil-O-Cel in the powdered form is to a certain extent elastic, allowing variation in expansion and contraction between the firebrick and the shell, eliminating all danger of excessive mechanical strains being set up between the two.

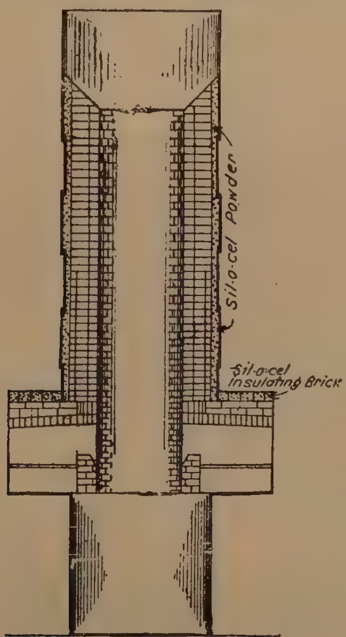

Fig. 6.—Lime Kiln Insulated with Sil-O-Cel Powder.

The insulation of lime kilns and similar equipment is shown in Fig. 6.

A unique application of the use of powdered Sil-O-Cel in annealing castings and other heat-treated metal forms is illustrated in Fig. 7, which shows an annealing pit partly filled with Sil-O-Cel powder, in which the castings are placed or suspended by chains until they are cooled to the proper degree for working. The annealing pit is built of brick and the depth of powdered Sil-O-Cel which is used is determined by the size and

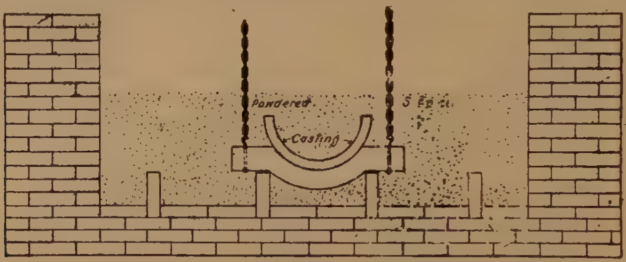

Fig. 7.—Annealing Pit Containing Powdered Sil-O-Cel for Slow Cooling of Castings.

shape of the castings to be annealed and the rate at which cooling is desired. This form of annealing eliminates to a very large extent the elaborate and costly annealing furnaces which are used in a great many plants and bids fair to become one of the most important uses of powdered Sil-O-Cel. This material has also been used as a packing material in boxes in which the metals to be heat treated are placed, the entire box being heated and allowed to cool slowly.

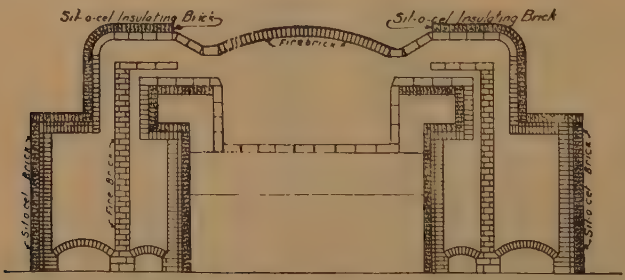

Fig. 8.—Glass Furnace Insulated with Sil-O-Cel Brick.

Glass furnaces insulated as shown in Fig. 8 are considerably more effective and economical than those operating without insulation.

Before leaving this subject, it would be well to again mention the fact that no attempt has been made in this presentation to cover all the fields of high-temperature work, but merely to outline the methods which are applicable to almost any form of high-temperature equipment.

Discussion

Lawrence Addicks, Douglas, Ariz.—It does not seem to me that the rôle played by carrying heat away from one surface to another has been brought out. A cellular construction prevents the motion of air currents. Also in the installation on the top of a reverberatory furnace, that Mr. Krieger mentions (in presenting the paper), where it stopped the cooling, there is besides the question of overheating. Another complex question, entirely apart from the necessity of not carrying a brick above a certain temperature, is that of not allowing acids and bases to come together at too high a temperature.

J. W. Beckman.—What cements together the individual cells in the insulating material? Also, how does the material behave in the presence of acids and alkalies, both strong and weak?

A. H. Krieger, New York, N. Y.—The material is formed or composed of myriads of lace-like skeleton remains of diatoms visible only under a powerful microscope. The diatom lives in salt and fresh water. On the Pacific Coast there must have been an abundant life of the diatom, because the beds, still in existence in Southern California, are fully 1,000 ft. through, and are hard and strong, standing a pressure of more than 400 lb. per square inch. The beds must have been many ages in forming and the diatoms have become firmly packed together. The samples shown are typical of the entire deposit as it was formed at the bottom of the sea. Once this structure has been destroyed, it is impossible to reconstruct it without artificial bonding materials. With the exception of hydrofluoric acid, acids do not affect this material. Strong alkalies dissolve it.

J. W. Richards, South Bethlehem, Pa.—This paper has interested me very much, and must interest all those who use furnaces and are concerned with saving heat. A general statement in connection with fuel-fed furnaces is that from 50 to 90 per cent. of the heat generated passes away with the gases from the furnace, while from 10 to 50 per cent. of the heat generated is lost in conduction or radiation from the walls. There is, then, the possibility of saving from 10 to 50 per cent. of the heat which is generated in the furnace by proper heat insulation of the walls. In electric furnaces, however, the problem is still more important. The gases which are generated in the electric furnace will carry out of the furnace only from 10 to 50 per cent. of the energy generated within the furnace, and the loss from radiation may amount to from 50 to 90 per cent. of the total energy put into the furnace. Therefore, the study and proper utilization of these heat-insulating materials concerns from 50 to 90 per cent. of the energy which is used for running electric furnaces,

and is one of the most important items upon which the furnace manager can experiment and study.

HENRY KREISINGER AND J. F. BARKLEY, Pittsburgh, Pa. (communication to the Secretary*).—At the experiment station of the Bureau of Mines in Pittsburgh, Pa., is a special furnace in the construction of which were used materials prepared from the Sil-O-Cel or infusorial earth, as it is commonly called. The furnace is a part of an apparatus designed for the study of heat transmission in steam boilers. The furnace is equipped

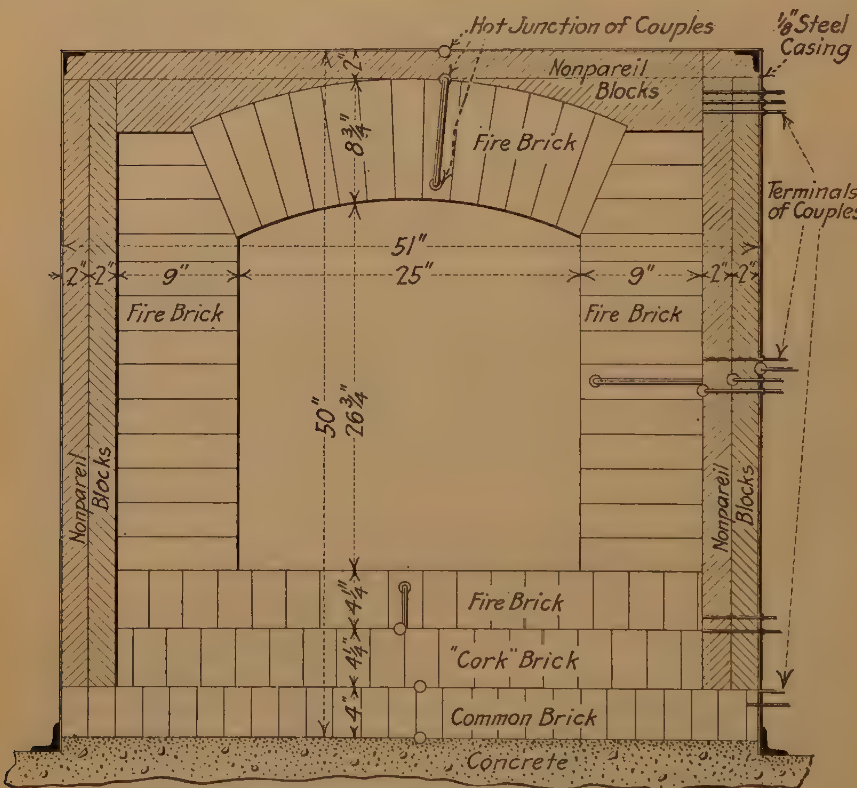

FIG. 1.—CONSTRUCTION OF FURNACE WALLS AND MATERIALS USED.

for burning natural gas, and on some of the tests a temperature of 2,400°F. is maintained. Fig. 1 shows the construction of the furnace walls and the materials used in the construction.

The material marked firebrick is a high-grade firebrick known by the trade name "Benezet" and made by the Harbison-Walker Refractories Co.

The material marked "cork brick" and the "Nonpareil block" are

* Received Sept. 16, 1915.

the product of the Armstrong Cork Co. The cork bricks are made of powdered infusorial earth, clay, and ground cork. When the brick is burned the cork burns out and leaves a red porous brick very much like ordinary red brick in appearance but much lighter. The main ingredients of nonpareil blocks are powdered infusorial earth and asbestos fiber. It

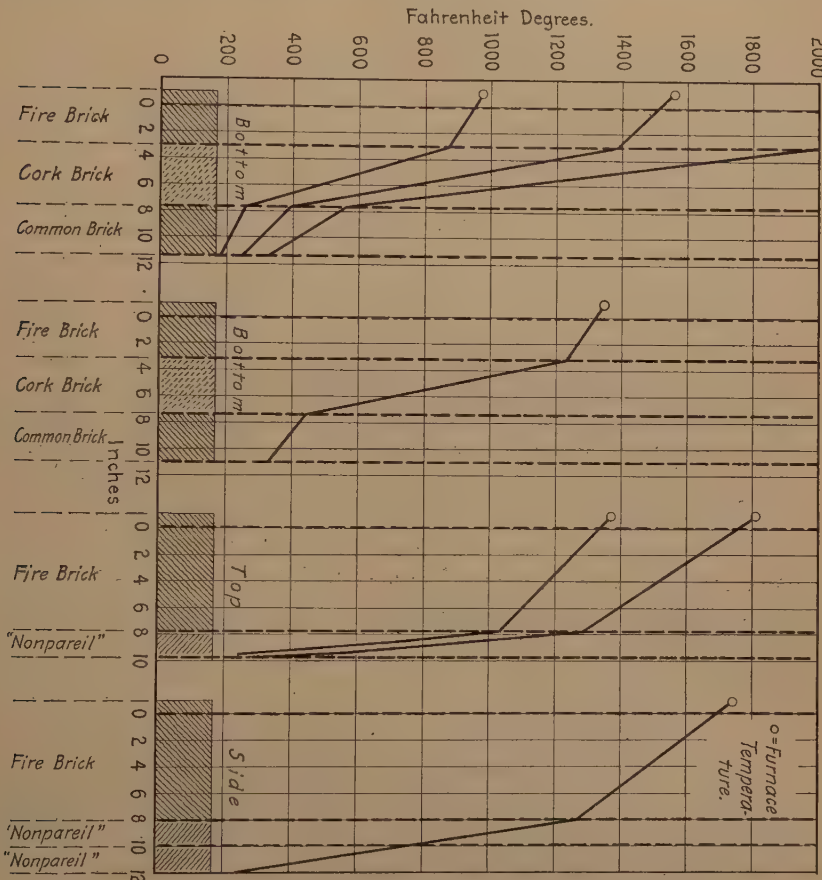

FIG. 2.—TEMPERATURE DROP THROUGH FURNACE WALLS.

is not burned and looks very much like the ordinary magnesia pipe covering. The part of the bottom marked common brick is made of common red brick.

As shown in Fig. 1, the walls are solid, without any air spaces, the latter having been found undesirable for high-temperature insulation. In order to obtain data for the computation of the quantity of heat dissipated from the furnace, also to obtain the figures of the comparative heat-insulating qualities of the different materials, the temperature drop through the walls was determined. For this purpose many thermocouples

were imbedded to various depths and at various places into the walls. The locations of the junctions of these couples are shown by the small circles in Fig. 1. The couples subjected to temperatures up to about 1,200°F. were of copper and constantan. For temperatures above this point, Hoskin thermocouples made of nickel and nickel-chrome were used. All readings were taken with a Leads Northrup portable potentiometer.

Fig. 2 shows the temperature fall through the bottom, through the top, and through the side of the furnace. The drop through the bottom is shown for four different furnace temperatures. The drop through the top is shown for two furnace temperatures, while the drop through the side for one furnace temperature only. As shown in Fig. 2 the temperature drop through the cork brick and through the nonpareil blocks is several times as great as the temperature drop through the firebrick or common red brick. This great difference in the temperature drop indicates that the cork brick and the nonpareil blocks are much better heat insulators than firebrick or red brick. By using about 4 in. of cork brick or nonpareil blocks in the construction, a 12-in. wall is made as good a heat insulator as a 30-in. wall made entirely of brick or containing air spaces. The advantage of the 12-in. wall constructed as shown in Fig. 1 is that it is cheaper and occupies less space than a 30-in. wall built of solid firebrick. Another advantage of the 12-in. wall, especially for experimental purposes, is that its heat capacity is much smaller, and therefore equilibrium conditions are reached in a much shorter time than would be the case with the 30-in. firebrick wall. This is of great importance in the investigations for which this furnace is used as it saves much time in heating up the apparatus. The disadvantages of the 12-in. wall for some types of apparatus might be its smaller strength.

The heat conductivities at various temperatures of the materials used in the construction of the furnace are given in the curves of Fig. 3. The abscissas are temperatures at which the conductivities were obtained; the ordinates are the conductivities. Attention is called to the fact that there are two ordinate scales on the chart; the lower scale applies to the nonpareil block and the cork brick; the upper high scale applies to the firebrick and common brick. The curve for the nonpareil blocks is plotted on absolute values obtained at the research laboratory of the Westinghouse Electric & Manufacturing Co. at Pittsburgh, Pa. The conductivity of the cork brick was computed from the data of Fig. 2 with conductivity of common brick assumed at 6.4 as a basis. This value of 6.4 was found to be the average of the results of various investigators. The curve of the cork brick is only relative to the assumed value of common brick; the curve may be either too high or too low, depending on whether the assumed conductivity of the common brick was too high or too low.

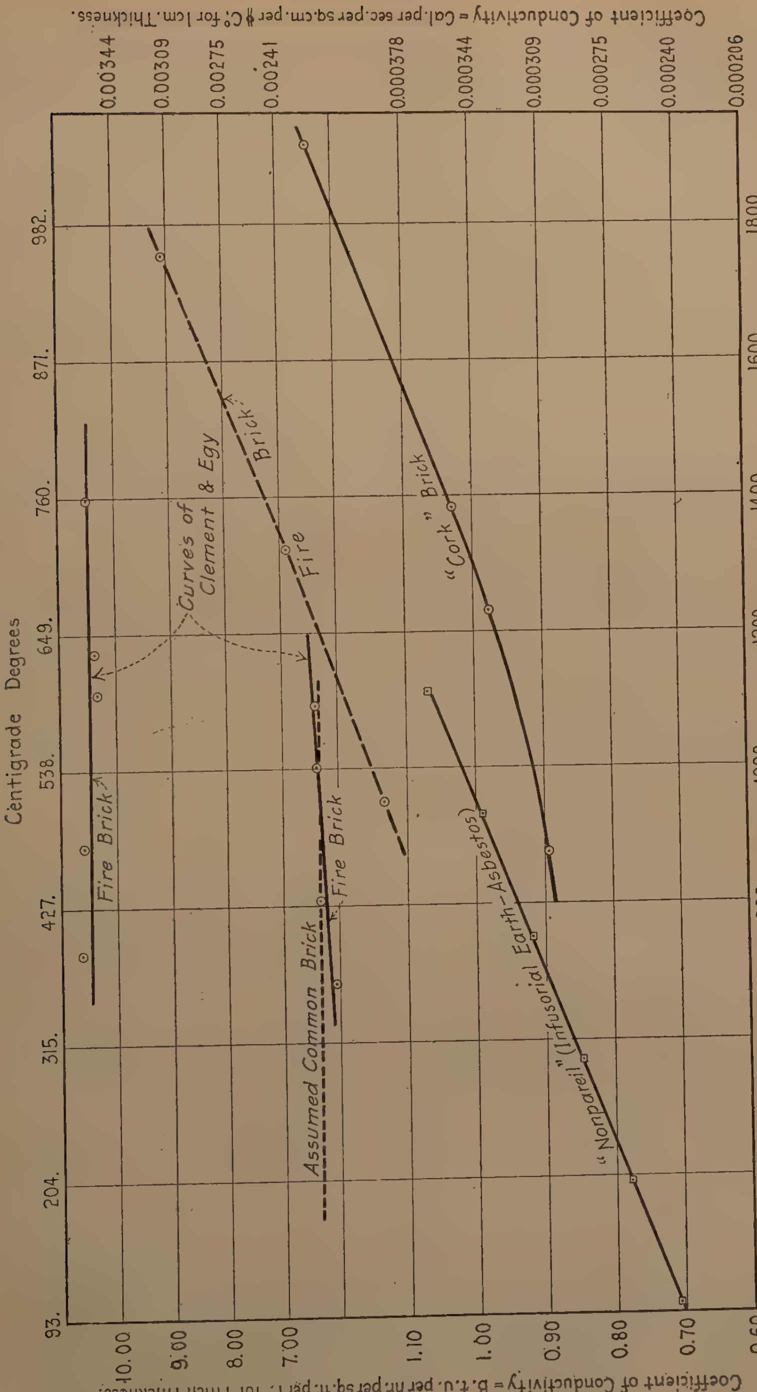

FIG. 3.—HEAT CONDUCTIVITIES OF DIFFERENT MATERIALS USED IN FURNACE CONSTRUCTION.

The curve of the conductivity of the firebrick was obtained from the data of Fig. 2 and is also only relative to the assumed conductivity of common red brick. The conductivity of firebrick computed from Fig. 2 varies greatly and the curve shown in Fig. 3 is only rough approximation. There is perhaps no exact value that will apply to one particular grade of firebrick; each individual brick has its own conductivity, differing perhaps many per cent. from other bricks of the same grade. To show the great variation in the conductivity of different firebrick the two curves of Clement and Egy are added to Fig. 3. The two curves were taken from a bulletin of the Engineering Experiment Station of the University of Illinois. According to the author's description, the firebrick test piece with which the lower curve was obtained was of reddish-brown color and of medium coarse structure. It contained very small pieces of white gravel throughout the mass. The higher curve was obtained with a test piece a little coarser in structure and of brown color. It contained a very small amount of gravel. As indicated by the two curves, the conductivity of one piece was about twice as high as that of the other.

The highest temperature maintained in the furnace is far below the melting of the firebrick lining so that it was not necessary to take any precautions on that account. In furnaces where much higher temperatures are maintained, good insulation may prove destructive to the firepoof lining. This is particularly true if the furnace gases carry slag in suspension. The slag settles on the furnace lining and causes the firebrick to melt at a somewhat lower temperature than it would if no slag were present.

The brick used for lining furnaces are usually mixtures of several compounds and therefore have no definite melting points. Some of the compounds melt at lower temperatures than others, so that the change from the first signs of softening to complete melting usually covers a wide temperature range. The following are the melting points of several kinds of brick used in furnace linings. The values are taken from Bulletin No. 10 of the Bureau of Standards, Melting Points of Firebrick, by C. W. Kanolt. The author defines the melting points as the lowest temperature at which a small piece of the firebrick could be distinctly seen to flow.

TABLE I.—*Melting Points of Various Bricks*

Name of Brick	Melting Point °F.		
	Low	High	Average
Fire clay	2,831	3,137	3,002
Bauxite	2,849	3,245	3,080
Silica	3,092	3,101	3,090
Chromite	....	....	3,722
Magnesia	....	....	3,929

The Commercial Production of Sound, Homogeneous Steel Ingots and Blooms

BY EMIL GATHMANN,* BALTIMORE, MD.

(San Francisco Meeting, September, 1915)

Through wide experience at numerous mills in the United States I have found that there is a decided difference of opinion among the producers of steel as to what constitutes commercially sound steel. Some metallurgists insist upon having, for certain grades of steel, a so-called "open steel," *i.e.*, one which contains numerous blowholes of varying sizes, and consequently but little volume of true pipe, although a central intermittent shrinkage cavity often extends well down into the ingot. Such blowholes and pipe are expected to weld up during rolling or forging, because their surfaces are not oxidized. Even if this be admitted, included slag particles and a high degree of segregation are bound to be present in the product of such ingots. Blowholes are the result of an oxidized heat, and all heats, unless made or finished in the electric furnace, are oxidized more or less. Subsequent deoxidation, or rather degasification, either in the ladle or in the mold, or in both, is necessary to reduce or prevent blowholes. As is well known, a greater yield of billets or sheets is secured from the ingot by allowing the formation of blowholes, but this is undoubtedly obtained at the expense of the quality of the product. Decided segregation of carbon, phosphorus, and sulphur, as well as small included slag particles, undoubtedly exists throughout the greater portion of all so-called open steel, and cannot be removed by cropping.

The first requisite for sound, homogeneous steel ingots and blooms is therefore, in my opinion, so to treat the steel in the furnace, ladle, and mold that "piping" steel is produced. The line of demarcation between harmful and so-called "harmless" blowholes is practically impossible to define. Is it not, therefore, the better and safer practice to use means for the elimination of blowholes and correspondingly reduce the segregation and allow the formation of a well-defined shrinkage cavity or pipe at the upper end of the ingot? With the relatively cheap deoxidizers available at the present time, *e.g.*, ferro-silicon, aluminum and titanium, there is no commercial reason why all steel should not be thoroughly degasified.

A degasified steel being provided, the method of freezing or solidifying the liquid steel into an ingot, and the subsequent working into blooms and thence into various products, is of great importance in reducing the crop or scrap portion of the steel, due to segregation and pipe. How to ac-

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* Manager, Gathmann Engineering Co.

complish this reduction at a minimum expense and without upsetting the administrative mill practice of present plants is the problem. It has been my experience that the solidifying of an ingot made from steel which has been practically deoxidized or "killed" in the ladle, or in the mold, depends upon the shape of the horizontal cross-section of the ingot at its various planes from top to bottom, and also upon the thickness and consequent heat-absorptive power of the various parts of the mold walls. An ingot with its larger horizontal cross-sectional area at the top is without question the best shape for obtaining the important "lag" in solidification of the steel, and whenever such large-end-up ingot can be conven-

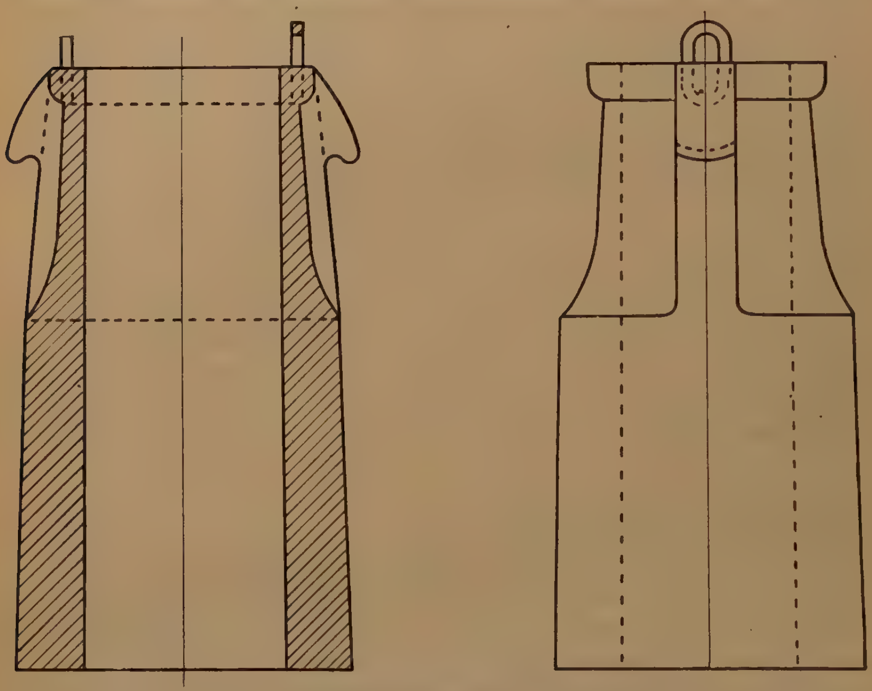

Fig. 1.—Gathmann Design of Ingot Mold.

iently used it is the best practice to do so. Nearly all of our large mills are, however, so equipped for handling and stripping the ingots that it is practically impossible to do this without extensive changes in equipment.

It is, therefore, necessary to employ means in ordinary big-end-down ingots to greatly accelerate the freezing and solidifying of the lower and middle portion of the ingot and thus provide liquid metal to compensate for the decrease in volume caused by the contraction during solidification of the ingot. This I have accomplished by giving the lower 70 to 80 per cent. of the metallic mold in which the ingot is cast a much greater degree of heat-absorptive capacity than the upper 30 or 20 per cent. thereof. At first I attempted to accomplish this by ribbing the lower exterior sur-

face of the mold in order to provide increased radiating area, or by water or air cooling the lower and middle portions of the mold. These earlier constructions were, however, found not to be practicable for commercial use. The desired results in differential cooling were then obtained by greatly thickening the lower and middle portions of the mold walls, consequently increasing their heat-absorptive capacity, and making the upper mold wall thinner and less heat-absorptive. This construction will be best understood by referring to Fig. 1, showing a general design of mold, and Fig. 2, showing a train of molds of this type. Many hundreds of molds of this kind have been in daily use under usual furnace and mill conditions during the past year. Designs of molds have been made and established in actual practice for from 1- to 10-ton ingots.

FIG. 2.—TRAIN OF GATHMANN 20 BY 24 BY 80 IN. INGOT MOLDS.

In order to insure an ample supply of liquid steel to compensate for the shrinkage in the upper portion of the ingot in very dense, quick-setting steel, it was found advisable to use walls of a material of poor heat conductivity in the uppermost portion of the mold to supplement the differential effect obtained by the combination of the heavy and thin mold walls; in other words, to provide a sink-head.

The usual type of sink-head resting on top of the mold, or rigidly secured in the upper part of the mold cavity, was found not to fulfill all the conditions necessary in a commercial sense, primarily because of the high cost of such sink-heads when efficient. Sink-heads of this kind must be of sufficient thickness and strength to hold the liquid steel without cracking or breaking. Even a slight crack or break in the sink-head is suffi-

cient to form a fin of steel and hang the ingot during its vertical shrinkage, thus causing surface cracks and defects in the body thereof.

Fig. 3 shows a method of freely suspending within the mold cavity a sink-head made of poor heat-conducting material. This type of sink-head, in conjunction with the heavy-wall mold, has given most excellent results, and is in regular use by several of the largest high-quality steel works in this country, in conjunction with the mold having walls with a heavy body and thin top. Such a mold, if properly designed, will produce a sound homogeneous ingot with about a 20 per cent. top crop, which is about 15 per cent. less than is usually necessary in ingots of like grade of steel

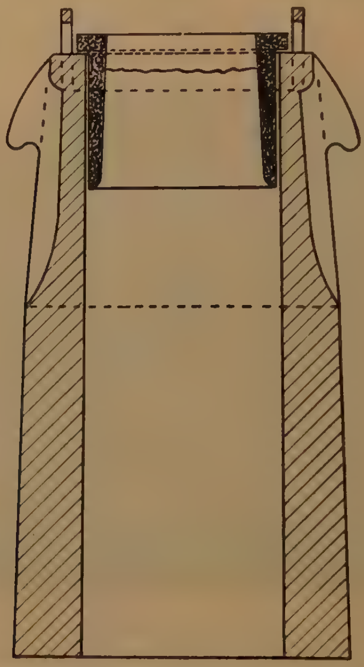

Fig. 3.—Sink-Head Applied to Gathmann Ingot Mold.

made in the old-type mold. Figs. 4 and 5 show photographs of two 2-ton split ingots of 0.85 per cent. carbon, dead-killed steel, made in the same group, bottom cast, in Gathmann and in the old-type molds respectively. Split ingots of vanadium-nickel-chrome steel, one cast in a mold of the standard type and the other cast in a Gathmann-type mold, are shown in Figs. 6 and 7.

With a sink-head in combination with the type of mold described the crop necessary to eliminate pipe and segregation is approximately 10 to 12 per cent. of the cold ingot.

When the ingots are allowed to solidify in the mold the pipe will be formed to its maximum depth. I therefore advocate a fairly early

stripping and rolling of the ingot, the actual time of stripping being governed by the cross-section of the ingot. No expensive change in administrative practice is required in the production of ingots by this method, which is in use by many steel plants, and it may be readily incorporated in the practice of any of the large steel works without upsetting in any manner the casting or mill practice.

Fig. 4.—Split Ingot, made in Gathmann Mold. Fig. 5.—Split Ingot, made in Regular Type of Mold.

The physically homogeneous condition of the ingot is obviously of primary importance in producing sound blooms and finished products. The method of blooming such ingots, is, however, also of importance. It has been my experience that in some heating and rolling practice there

is danger at times of actually forming a false pipe, or rather fissure, in the ingot. This is especially the case where the ingot has been allowed to solidify and become cold in the mold. Many mills have found it good practice to strip the mold from the ingot before its central portions have entirely solidified and place it immediately in the soaking pit or heating

FIG. 6.—STANDARD INGOT OF VANADIUM-NICKEL-CHROME STEEL.

FIG. 7.—GATHMANN INGOT OF VANADIUM-NICKEL-CHROME STEEL.

furnace in a vertical position, allowing the temperature to become equalized and the final feeding, due to shrinkage, to occur while in the soaking pit. The ingot is then rolled while the interior is hotter and more plastic than the exterior. *Great care* must, however, be exercised that the ingots be allowed to remain in the soaking pit until the steel is entirely set, because rolling with any green steel in the ingot will produce spongy

centers. It is, therefore, necessary to determine the proper heating period before pulling specific sizes of ingots for blooming.

The best method of reducing ingots to blooms is without doubt to effect most of the reduction in butt-to-head passes, and I have seen a marked improvement in crop reduction obtained by this method of rolling, especially when the piped section was confined to the upper 20 per cent. of the ingot. Irrespective of any heating and mill practice which it may be desired to use at any specified mill, the physical condition of *cold ingots* split open for inspection is undoubtedly the true index to the value of any method for producing sound, homogeneous steel.

The theoretically ideal method of making sound, homogeneous steel is undoubtedly with the big-end-up mold, because the increasingly large area of the ingot toward the top compensates automatically for any ir-

FIG. 8.—ONE-TON INGOT, MADE IN GATHMANN BIG-END-UP MOLD.

regularities in the temperature of the steel or in the teeming practice. The method I propose for utilizing the big-end-up mold in car practice was fully described in my previous paper.[1] Fig. 8 shows a split ingot produced by my method under normal conditions with the big-end-up practice, the crop for piped section being approximately 5 per cent.

In the commercial production of sound ingots and blooms, tonnage and quality must, however, be jointly taken into consideration. I would, therefore, recommend in the usual mill practice where cars or bogies are used for carrying the molds, the heavy-walled body, light-walled top, big-end-down mold, as shown in Figs. 1 and 2 of this paper. I would also recommend the use of a suspended sink-head of non-conducting material,

[1] *Trans.*, xlv, 461 to 472 (1913).

one type of which is shown in Fig. 3, to fully compensate for the contraction in the upper portion of the ingot. The molds and methods as described are applicable for the production of any grade of steel cast in ingots, and will invariably improve the quality and lessen the top crop. Thousands of tons of these molds have been made of both direct and cupola Bessemer iron, and have given excellent service as to number of heats per mold. Reports from mills using these methods and molds show that an important saving has been effected thereby in reduction of crop ends, and in making good steel better, in all cases where the molten steel has first been made sound by thorough deoxidation, which latter I consider the prime essential for the production of commerically sound, homogeneous steel.

Discussion

H. W. Lash, Cleveland, O.—I have found in practical experience that it is not objectionable to have some blow-holes in ingots. They enlarge during solidification and have a tendency to cause the steel to rise. If they are of fairly small size they weld up quite readily, especially where they are some distance from the surface. In that way a pipe, which is much more detrimental than a few small blow-holes, is prevented. There is a very great difference in the formation of blow-holes in steel. You get one condition in soft steel and entirely another condition in rail carbon, or the still higher grades. The figures that are given in Mr. Gathmann's paper I imagine are taken from the higher carbons, as it is very seldom that piping occurs in soft steel.

J. W. Richards, South Bethlehem, Pa.—This contribution is very welcome, and I understand that the process of Mr. Gathmann is in practical use and has proven successful. I believe the experience of steel makers is that an ingot which has a pipe is of more solid structure, and therefore takes a great deal more work in the rolls and in mechanical treatment, than an ingot which has some blow-holes. The density of the piped steel is undoubtedly increased. My opinion is that it is undoubtedly best to try to completely deoxidize or kill the steel, and then try to overcome the secondary difficulties which are brought in by having a denser and more compact metal. If you completely deoxidize you know where you are at. If you leave some blow-holes in you may at times leave more than you desire, at others less. The blow-holes may come a little nearer to the surface of the ingot than you wish, and in reheating cause surface imperfections. I therefore think Mr. Gathmann's thesis is sound. But, in carrying it out in practice, you have to use some such device as Mr. Gathmann has invented to prevent a deep pipe which perhaps may go down to the center of the ingot. What he has succeeded in doing marks, I think, a distinct advance in the production of sound steel.

The Electric Furnace in the Foundry

BY WILLIAM G. KRANZ, M. E.,* SHARON, PA.

(San Francisco Meeting, September, 1915)

THE increased service demands on some of the products of the National Malleable Castings Co. prompted it about eight years ago to investigate the electric furnace, both in America and in Europe. The process had already been sufficiently developed in Europe to lead us to believe that the electric furnace would most nearly meet our requirements.

After further investigation and consultation with the highest authorities on the subject, we decided upon the Heroult type as the simplest and most practicable.

In 1910, we built a furnace of this type, of 300 lb. capacity, for experimental purposes, and although it was crudely constructed the quality of the material we were able to make justified the installation of a 6-ton Heroult furnace in 1912.

This furnace has three electrodes, 17 in. in diameter, uses three-phase, 60-cycle current, and is equipped with Thury regulators, which give us good regulation and a uniform load on the line even when cold melting. The power factor of the furnace is 94 per cent.

Since July, 1912, the furnace has been in continuous operation, producing up to the present time over 20,000 tons of both carbon and alloy steels of varying analyses and of exceptional quality. Both cold- and hot-metal charges have been used, with a power consumption of about 150 kw-hr. per ton in the case of hot-metal charges and from 500 to 600 kw-hr. per ton in the case of cold charges.

The electric energy consumed varies according to the final analysis of the steel and the amount of refining required. Too much cannot be said about the refining possibilities of the electric furnace, for no other method of steel manufacture can compete with the electric in this respect.

In proper operation lies the whole secret of success. That steel has been made electrically means nothing, for when the furnace is operated under oxidizing conditions the quality is no better than that of open-hearth material; but when properly made, electric steel is as good as

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* Vice-President, the National Malleable Castings Co.

that made in the crucible, and very much less expensive. The operation of the furnace is simple and the resultant composition is scientifically accurate. I might cite an example in this connection:

On a heat of steel treated under a slag of high silica content some metallic aluminum was added in order to reduce the silicon from the slag in accordance with the following chemical equation: $3SiO_2 + 4Al = 3Si + 2Al_2O_3$. Upon final analysis it was found that the amount reduced was theoretically correct in accordance with the above equation.

The most important of the many advantages of electric steel castings over those made by the ordinary processes are briefly summarized below, and it should be kept in mind that these apply not only to carbon steels but to the alloy steels as well.

1. Absence of segregation, elimination of oxides, and absolute uniformity of composition regardless of atmospheric conditions which affect open-hearth furnaces.

2. Almost entire elimination of sulphur is possible (an important consideration in steel castings) and complete control of the other elements.

3. Great tenacity, giving ability to withstand much more abuse and fatigue without rupture.

4. High ratio of elastic limit to ultimate strength.

5. A more ready response to heat treatment and with much more uniform results.

6. Perfect control of pouring temperature, combined with ability to obtain very hot metal, so that light and intricate shapes are readily cast.

Let us take up the advantages of the more important of these qualities of electric steel, and investigate them more fully.

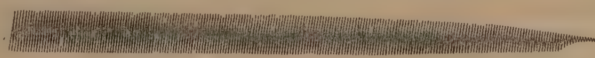
First.—Absence of segregation and oxides has been firmly established by a great many investigations in our chemical and microscopical laboratories. As a more practical proof of the elimination of oxides, we know that additions of any of the ferro-alloys to the bath will be found alloyed with the steel in their theoretical amounts. This is even the case with elements, such as aluminum and titanium, which are so susceptible to oxidation, proving conclusively in our minds that there could be no oxygen present in the steel. Leading authorities have agreed for some time that the electric steel furnace is the one means of preventing segregation.

W. R. Walker, in May, 1912, read a paper on Electric Steel Rails before the American Iron and Steel Institute, in which he stated that "Ingots of even eight tons had been produced electrically which were practically free from segregation."

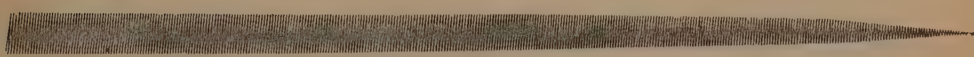
Second.—We know of no authority who doubts the practical elimination of sulphur in electric-furnace operation. We have repeatedly

reduced the sulphur to a trace in irons containing as high as 0.30 sulphur. The rapidity with which this reduction takes place depends somewhat upon the carbon content of the material, the sulphur reducing very much more rapidly in the higher carbon materials.

Third.—To illustrate the great tenacity of electric steel we submit (Fig. 1) exact reproductions (reduced) from two tests of two similarly treated specimens of steel, which are typical of a large number made on an Upton-Lewis toughness testing machine. One of these is electric and the other open-hearth, of almost identical analysis, as shown below. The marked superiority of the electric steel specimen, especially as to its tenacity, is shown in the following tabulations:



Basic open-hearth steel (annealed). 213 cycles.



Electric steel (annealed). 345 cycles.

FIG. 1.—FATIGUE TESTS.

Electric Steel

C, 0.24; Mn, 0.52; Si, 0.25; P, 0.010; S, 0.019.

Elastic limit	36,400 lb.
Ultimate strength	65,300 lb.
Elongation	36 per cent.
Reduction	55 per cent.
Toughness test (fatigue) to break	345 cycles

Basic Open-hearth Steel

C, 0.23; Mn, 0.53; Si, 0.24; P, 0.011; S, 0.038.

Elastic limit	34,800 lb.
Ultimate strength	63,000 lb.
Elongation	29½ per cent.
Reduction	35 per cent.
Toughness test (fatigue) to break	213 cycles

Fifth.—During the past year we have developed an electric steel having remarkable physical qualities after heat treatment. This was accomplished without resorting to any of the high-priced alloys. The steel, when subjected to shock or static pull, will stand from four to five times as much stress without distortion as the ordinary open-hearth product. This assertion is not based on a few tests, but on over 3,000 made up to the present time.

We have made a great many similar tests of heat-treated open-hearth material, and have invariably found that in certain specimens

the physical properties are impaired by the treatment rather than benefited. This is not, however, the case with the electric-furnace product. The old saying that "In order to make good bread you must start with good dough," we believe applies to the steel industry as well.

Conclusions

The greatest advantage of the electric-furnace process over all others is its uniformity of product. The open hearth under certain conditions will, we know, produce steel of very good quality, but there are so many contingencies, such as atmospheric conditions, stack draft, fuel and furnace variations, beyond the control of the operator, that absolutely uniform results are impossible.

Electric-furnace products are looked upon by some with skepticism, due to those who have adopted the process thinking it a panacea for all of their ills, and who have furnished the trade with products not worthy of the name. The lack of knowledge and the inexperience of the operator should in no way condemn the process. The production of perfect castings does not entirely depend upon the quality of the material of which they are made, but upon the foundry practice to a very great extent. Frequently excellent material is ruined by bad foundry practice. Nevertheless we feel that the electric furnace, with its perfect control of composition and temperature, fills a long-felt want in the industry.

Discussion

M. Petinot, Niagara Falls, N. Y. (communication to the Secretary*). —I have read this paper with considerable interest and agree perfectly with Mr. Kranz that steel for castings made in an electric furnace is not necessarily, because of that fact, all good steel, and also that the qualities which he mentions, such as absence of segregation, elimination of sulphur, great tenacity, etc., can be obtained in an electric furnace if operated under proper metallurgical conditions.

An unusually low content of sulphur in steel for castings is not, of course, an absolute necessity and the range of this element usually obtained in steel made in basic open-hearth furnaces is sufficiently low to meet all physical requirements.

The main factor in making steel for castings, regardless of the process, is the complete deoxidation of the bath to prevent the formation of segregation and also increase the physical qualities of the steel. Such complete deoxidation is accomplished in basic electric furnaces by holding the metal sufficiently long in the presence of a blanket of carbide of calcium, as in the Heroult or Stassano process, or by the addition of

* Received Sept. 13, 1915.

50 per cent. ferrosilicon, as in the Girod or induction furnace process. However, if steel is so completely deoxidized in the furnace that when, as pointed out by Mr. Kranz, an addition of aluminum or titanium is made the added metal will not oxidize but will be found in the metallic state in the steel, there can be no doubt that such steel will reoxidize during the tapping of the furnace and also during the teeming from the ladle into the molds. While Mr. Kranz has undoubtedly through inadvertence failed to mention the fact, it is probably the practice at his plant to add aluminum in the ladle during the tapping of the furnace to completely deoxidize the steel.

Even admitting that an electric furnace properly controlled must give a more completely deoxidized product in the furnace than it is possible to obtain in such oxidizing processes as open hearth and converter, I still question its great advantage if it is still necessary to complete the deoxidation in the ladle.

In the accompanying schedule I have reproduced Table No. 1 showing the comparative properties of electric and basic open-hearth steel as printed in Mr. Kranz's paper, and also the results of samples from six heats of basic open-hearth and converter steels of approximately the same chemical composition.

These six analyses are representative of the quality of thoroughly deoxidized steel produced under good foundry practice and it will be noted that the physical properties compare very satisfactorily with those of the electric steel reported by Mr. Kranz.

Mr. Kranz mentions that "absence of segregation and oxides has been firmly established by a great many investigations in our chemical and physical laboratories" and gives as a proof of this fact that when ferrotitanium or aluminum is added to their electric steel either of these metals will be found as such in the steel, indicating that no oxides were present which would cause their oxidation.

I am inclined to believe that many metallurgists will be interested in obtaining full details in reference to this particular point made by Mr. Kranz. Personally, I have made hundreds of heats of steel in the same type of furnace which he is using. In many of these heats 0.10 per cent. titanium was used and in no case have overtraces of titanium been found in the finished casting.

At other times I have used aluminum in the ladle and occasionally added it into the stream of steel as it flowed from the ladle into the molds, particularly at the end of the operation because this addition of aluminum was judged necessary to insure the soundness of the castings. In no case has aluminum been found in the finished casting, but in every case where an examination has been made for the presence of alumina, the oxide of aluminum has been established both by chemical and microscopical means.

I am of the opinion that what Mr. Kranz has reported as aluminum in the steels, which he has treated with that metal, is in reality alumina, and I believe that more complete information on this particular point will interest metallurgists generally.

TABLE I

	Elastic Limit	Ultimate Strength	Per Cent. Elong.	Per Cent. Red.	C	Mn	Si	S	P	
Electric steel......	36,400	65,300	36.0	55.0	0.24	0.52	0.25	0.019	0.010	Referred to by Mr. Kranz.
O. H. basic........	34,800	63,000	29.5	35.0	0.23	0.53	0.24	0.038	0.011	
O. H. basic........	40,900	70,400	35.9	56.3	0.24	0.55	0.30	0.020	0.020	
O. H. basic........	36,200	64,200	35.9	61.3	0.23	0.51	0.35	0.020	0.012	
O. H. basic........	35,800	68,300	35.9	61.4	0.23	0.77	0.34	0.027	0.012	Test piece unannealed.
O. H. basic........	38,700	61,600	35.0	54.0	0.19	0.56	0.22	0.027	0.014	
Converter steel....	38,000	69,200	32.0	53.6	0.21	0.76	0.37	0.055	0.041	
Converter steel....	40,500	71,000	32.0	52.8	0.21	0.68	0.34	0.052	0.044	

Carl H. Booth, Chicago, Ill.—Some people dwell a great deal on the superior quality of electric metal. I have believed that it is practical to make as good a quality of steel in other types of steel-melting equipment, providing skilled handling is obtained, but present-day experience indicates that the electric furnace is coming into general use because of the cost factor. The fact that you can get increased temperatures in an electric furnace makes it practical to make very thin castings, very nice delicate ones; it is ideal for small castings. Those are features which make it a highly attractive form of melting equipment, but the principal point and the one that will decide finally for those who are making steel and want to adopt some other form of melting, will be the cost of the manufacture of the steel itself. With other types of melting equipment you can get steel equally good. It will cost more to do so. I think that one point ought to be brought out here. It is well known that in Europe most of the high-grade steels are made in electric furnaces. That has probably led to the assumption that electric product is always better. As an argument for the salesman it is very nice to be able to talk that way. We really believe that it is easier to make a good metal with the electric furnace, but we do not say that it is not equally possible to make as good with other types of furnaces. The commercial question is the cost. With electric furnaces of proper design the cost can be made lower than the cost of melting steel with fuel furnaces, and it is this lower cost of steel production that is forcing the installation of electric steel furnaces.

J. W. Richards, South Bethlehem, Pa.—We should not overlook the fact that there is a type of material that can be made by the electric furnace which cannot be made by the other. I refer to a quality of steel which is intermediate between open-hearth steel and crucible steel. In practice there is a big gap in the quality of those two materials, and also

in the price. It is possible to produce with an electric furnace, at an intermediate price, an intermediate quality of steel which fills a new field.

I have visited many steel casting plants in Europe, particularly in Norway and Sweden, and I am convinced that Mr. Kranz's statement properly represents the possibilities of the electric furnace. I think Mr. Booth is too modest in saying that you can obtain the best qualities of steel by other ways. The electric furnace competes with the very best quality of steel which is being made in the crucible. There are numerous plants in Germany that have torn out their crucible equipment and have substituted the electric furnace. Old established crucible steel plants there are making their finest qualities of steel, alloy steel and carbon steel, with the electric furnace. It is all a matter of price.

H. W. Lash, Cleveland, O.—I have always understood that the electric furnace offered advantages in the way of purity. In the open-hearth furnace there is more or less sulphur to deal with in the fuel, and of course oxygen is always present.

I have talked with Mr. Petinot and he showed me many results of electric-made steel where they had almost eliminated the phosphorus acid sulphur and produced a steel that was very low in manganese and still worked well in the forge, largely because of the absence of oxygen.

Carl H. Booth.—My thought is that if you are willing to spend money enough you can take almost any furnace, whether electric or not, and the man who knows how could make steel, and make it good. Each different type of furnace has its advantages and disadvantages. The open hearth especially has disadvantages because of the conditions in the furnace. The electric furnace has a big advantage because of its ideal conditions, conditions which are under your control at all times. The atmospheric conditions are so good that you can do almost anything you want to do. Our thought has been to make the electric furnace so simple and so efficient that the cost of making these different steels will be cheaper than those made in any other way. The real advantage after all that will contribute in making the use of the electric furnace more general than the use of other types of equipment will be not only that it can make a good steel—it will have to compete on that basis just the same—but the question will be as to whether or not it can make steel cheaper than any other way. The earlier types of electric furnaces were so expensive to operate that they were largely used in the manufacture of quality steel, principally high-grade alloy steels, for which the electric furnace is ideally suited, and for which a high price could be obtained. In Europe today the electric furnace is being used extensively for this type of service in place of crucible melting. Only in recent years, however, have electric furnaces been adopted for the manufacture of the cheaper

grades of steel. There are electric furnaces running today in the United States which are producing steel at a lower cost than can be obtained with any other type of melting equipment for the same daily tonnage.

J. W. RICHARDS.—There is one point which needs to be cleared up. The crucible process can make the highest quality of steel, equal, let us say, to the best made in the electric furnace, but it requires the highest quality of raw material. It is possible for the electric furnace to make steel of that same quality from impure raw materials which cannot be used in the crucible at all. You cannot refine in the crucible. You can refine and make the highest quality of steel in the electric furnace, from what crucible steel makers would call low-grade material. That fact alone has sounded the passing of the crucible process. At the present time I believe there are 35 electric steel furnaces under construction in the United States, and many more in operation. The United States Steel Corporation, for example, is putting up for one firm a plant of eight electric furnaces. The principal increase in use of the electric furnace at the present time is based on its superiority for making steel for shrapnel and for munitions of war. I understand that electric steel has been found far more reliable and suitable than ordinary steel for these purposes in Europe.

The Duplex Process of Steel Manufacture at the Maryland Steel Works

BY F. F. LINES,* SPARROWS POINT, MD.

(San Francisco Meeting, September, 1915)

It is not the intention of the writer to enter into a discussion of the relative merits of the duplex process as compared with the straight scrap and pig iron process, working under the same conditions, but rather to present in a general way several modifications of the former process, and then describe the method adopted by the Maryland Steel Co., as best suited to local conditions, and also to meet the metallurgical problem presented in the removal from the iron of an unusual constituent—chromium.

The principal factors to be considered in deciding whether or not duplexing should be employed are:

(1) Chemical constituents of pig iron.
(2) Variety and quantity of steel to be produced.
(3) Relative cost of pig iron and scrap.
(4) Regularity of operation.
(5) Capital to be invested.
(6) An existing Bessemer plant.

Having decided to adopt the duplex process, several combinations may be used, viz.:

(1) Acid-lined converter with basic open hearth.
(2) Acid-lined converter with electric furnace.
(3) Basic-lined converter with basic open hearth.
(4) Basic-lined converter with electric furnace.
(5) Basic open hearth with electric furnace.

The combination of an acid-lined converter with a basic open-hearth furnace is the one most commonly employed, and as it is the one best adapted to the conditions encountered at the plant of the Maryland Steel Co., we will briefly outline the advantages and disadvantages of the different methods of working.

There are two general methods of blowing, as follows:

(1) The charge is desiliconized and partly decarbonized in the converter. On account of the difficulty in regulating the percentage of car-

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bon in the blown metal, this method is not desirable where the amount of phosphorus or other impurities to be removed is so low that the duration of the subsequent open-hearth operation is determined by the time required to adjust the carbon to the specification limits. This objection, however, does not hold when the blown metal is to be added to a part charge of pig and scrap previously melted in the open hearth, or when it is to be treated by the continuous process in furnaces of 200 to 300 tons capacity.

(2) The converter metal is completely desiliconized and the carbon blown down to about 0.10. This method of blowing commends itself on account of the regularity of the carbon content of the metal going into the open-hearth furnace, and in certain cases the more complete removal of other constituents.

Two methods of charging the open-hearth furnaces have been employed, dependent upon the facilities for handling the blown metal. In the first plants built it was necessary to send individual Bessemer blows to the open hearth. However, in the plants built recently it has been found advisable to assemble the blows necessary for an entire open-hearth heat in one ladle in the converting department. This method requires larger crane capacity, but has the great advantage of reducing the charging of the furnace to one operation. Both of these methods allow the slag to be kept out of the furnace. In deciding upon the most desirable size of an open-hearth furnace, the question immediately arises as to the type—tilting or stationary; and as this discussion is limited to duplexing, where 8 to 12 heats are tapped in 24 hr., there is little to be said in favor of the stationary furnace. Of late the tendency has been toward the installation of large tilting furnaces of 200 to 300 tons capacity, and while much of the evidence is contradictory, it can be safely stated that these larger furnaces show a saving in the cost of operation and repairs over the smaller ones. We are indebted to Dr. Schuster for a valuable paper read at the May, 1914, meeting of the British Iron and Steel Institute, in which he published the results obtained when working with different sizes and types of furnaces under identical conditions. As a result of experiments, covering a working period of 12 months, he has been able to establish conclusively that:

(1) The quality of steel produced is both physically and chemically independent of the type of furnace employed.

(2) The yield is practically the same whenever the same conditions are observed in furnaces of different types.

(3) Although the actual first cost of the 200-ton furnace is higher than the cost of the tilting furnace of smaller capacity, or of a stationary open-hearth furnace, this cost, calculated on the daily tonnage produced, works out most favorably for the 200-ton furnace.

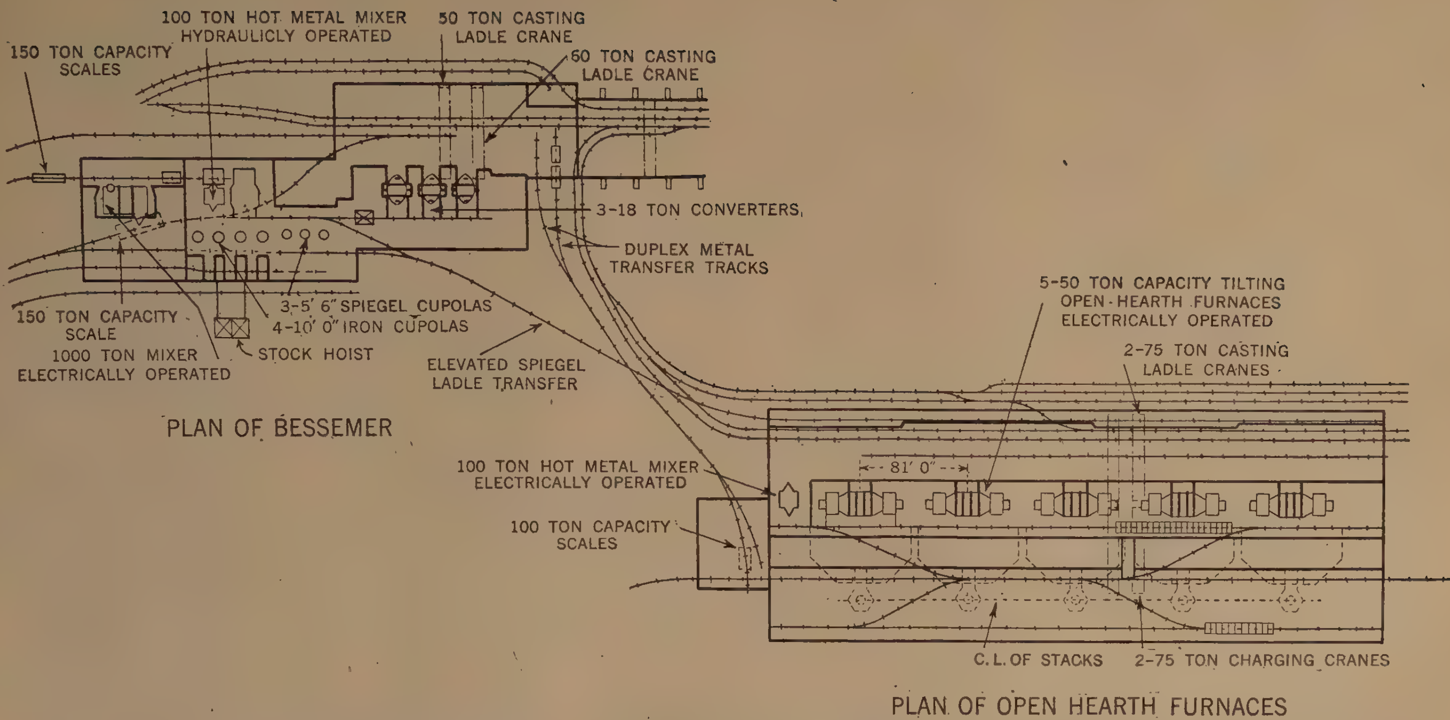

Fig. 1.—General Arrangement of Converter and Open-Hearth Departments of Maryland Steel Co., Sparrows Point, Md.

(4) The fuel consumption is lower in the 200-ton furnace than in the other furnaces.

(5) The life of the refractory lining is longer in the 200-ton furnace.

(6) The labor of operating is less in the 200-ton furnace.

At the plant to be described, the following points decided the adoption of the duplex process: This plant was originally built to manufacture Bessemer steel rails, the blast furnaces and steel department being proportioned accordingly; the soaking pits in the rail mill were designed to treat the steady stream of hot ingots produced by the Bessemer process. To be able to supply open-hearth steel in place of Bessemer steel, the duplex process was adopted, using five 50-ton tilting furnaces, emptied at each heat. It was also decided at the start to blow the iron to about 0.10 carbon and to assemble the converter charges required for an open-hearth heat in a single ladle in the converting department.

The advantages of this plan for this plant are:

(1) Low first cost and no changes necessary in the existing plant.

(2) The ability to produce all Bessemer steel or all open-hearth steel, with very little change in the volume of finished product.

(3) The blast furnaces may be run to the best advantage, as no limit is set in regard to the silicon of the metal delivered to the converters.

(4) The change to duplex or to straight Bessemer steel is made without expense or loss of time.

(5) When working on open-hearth steel, the Bessemer department is fully employed.

The converting plant, built in 1889, the general arrangement of which with respect to the open hearth is shown in Fig. 1, is contained in one building. At the south end of the building is a 1,000-ton, cylindrical, electrically operated metal mixer. The iron from the blast furnaces comes to the mixer building in 45-ton ladles. A crane of 75 tons capacity, with an auxiliary 10-ton hoist for tipping the ladles, is used for handling the iron. The iron ladles coming from the blast furnaces are weighed on 150-ton standard-gage track scales. Coke-oven gas or fuel oil is used for heating the mixer, a burner being arranged at each end for this purpose. However, it has been found necessary to use only one burner and the mixer has been kept in excellent condition by the use of 100,000 cu. ft. of gas in 24 hr. The mixer has a 9-in. magnesite lining, backed up by 13 in. of fire-clay brick.

Directly under the pouring spout is another standard-gage track scale for weighing the metal charges for the converter. From the mixer, the molten metal is taken to the casting side of the converters. On its way it passes under the iron cupola runners, where it may be stopped to receive an addition of cupola metal. The converters are served by two traveling cranes, one with a 50-ton ladle hoist, having a 5-ton auxiliary

hoist for pouring the iron into the converters, and the other, a 75-ton crane which takes the blown metal from the converters.

The method of pouring the iron into the converters is shown in Fig. 2. The three converters are of the concentric type, of 18 tons rated capacity, but 20 to 22 tons of metal have been repeatedly blown. The converters, operated by hydraulic cylinders, are mounted on structural-steel columns and are 33-ft. centers. The converter bottoms have 24 tuyères, each tuyère having eight 9/16-in. holes, giving a total effective area of 47 sq. in. per bottom. With 21 tons of metal, the depth of the bath is 24 in. The blown metal from these three converters is received in a 60-ton transfer ladle, which is conveyed by a motor car to the open hearth, a distance of 350 ft. The ladle is carried to the proper furnace by one of the 75-ton charging cranes, and placed in position as shown in Fig. 3. This type of transfer ladle differs from those in general use for this purpose, in that the metal is not poured over the top of the ladle into the furnace, but is tapped through a 3-in. nozzle placed on the side.

The open-hearth building is 459 ft. long, and 138 ft. wide, there being a 65-ft. clearance on the pit side of the furnaces and 65 ft. on the charging side. On the latter side there are two 75-ton electric traveling ladle cranes for handling hot metal and duplex metal on the charging floor, and a 10-ton charging machine. A 120-ton metal mixer is placed in line with the furnaces, and is so arranged that the metal can be drawn out on the charging side as a "fill out" for dead soft duplex metal, or on the pit side for recarburizing the heats in the casting ladle. Scales are located on both sides of this mixer. On the tapping, or pit side, 75-ton ladles, handled by two 75-ton traveling cranes, are provided. The ladle cranes are supplied with two auxiliary hoists for handling the recarburizing metal. The pit slag is handled in steam-dumping ladles.

The gas-producer building runs parallel to the open-hearth building, and contains nine self-cleaning Hughes producers, as well as two dolomite kilns, and one crusher. The stockyard is located between the gas producers and the open-hearth building. The cold stock is loaded on a stock platform, adjacent to the charging floor, by a 10-ton magnet crane, which fills the boxes on the ground level from standard-gage cars.

The dimensions of the furnaces are:

Length between chill plates	31 ft. 2 in.
Length on the metal line	24 ft. 0 in.
Width on the metal line	11 ft. 6 in.

The actual area on the metal line is 175 sq. ft., when made up to hold 50 tons. This area generally averages 220 sq. ft. and with this 60-ton heats are tapped. The regenerators are:

Gas	8 ft. 3 in. by 22 ft. by 10 ft. 4 in., or 1,815 cu. ft.
Air	13 ft. 3 in. by 22 ft. by 10 ft. 4 in., or 2,915 cu. ft.

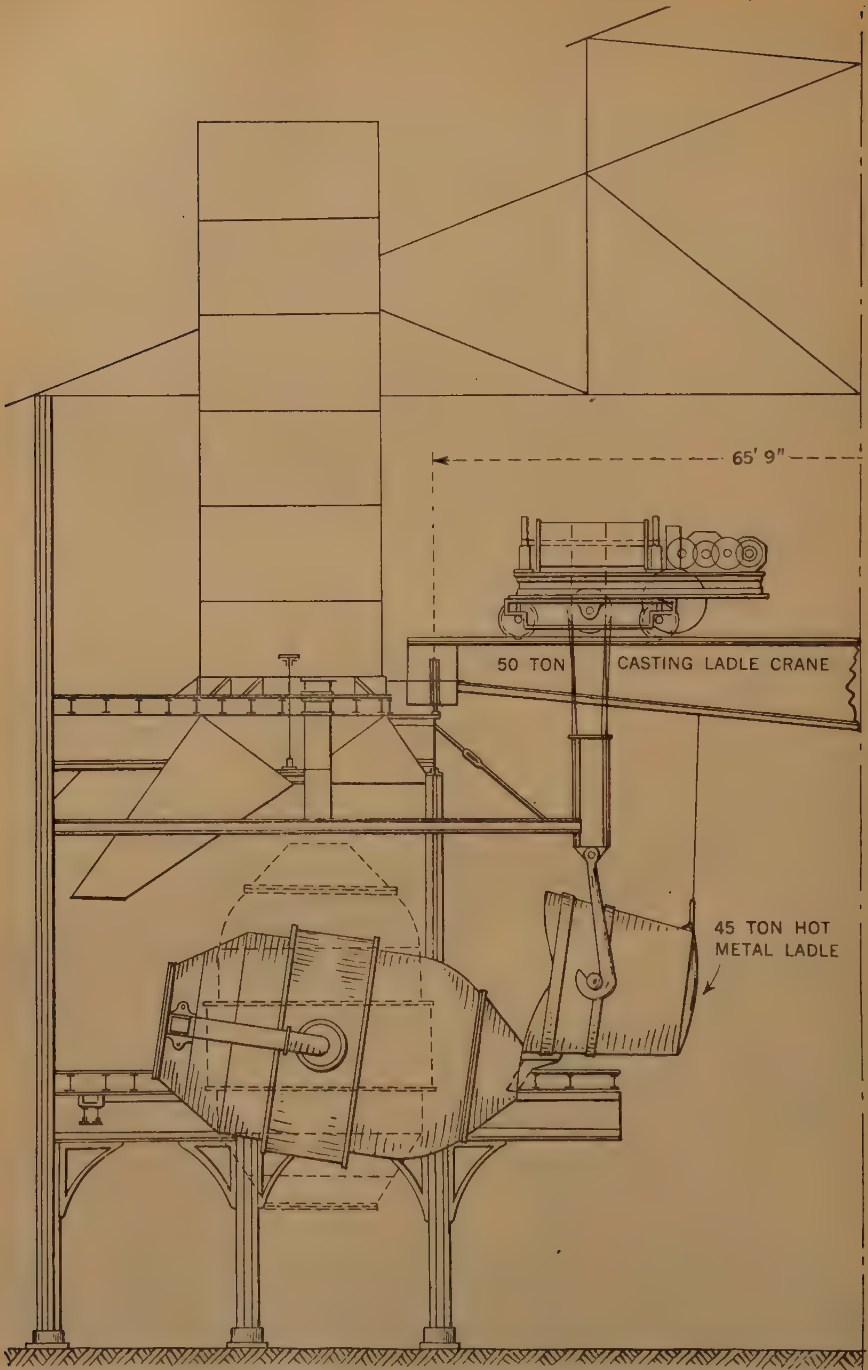

FIG. 2.—PART SECTION OF FIG. 1, SHOWING METHOD OF CHARGING THE CONVERTERS.

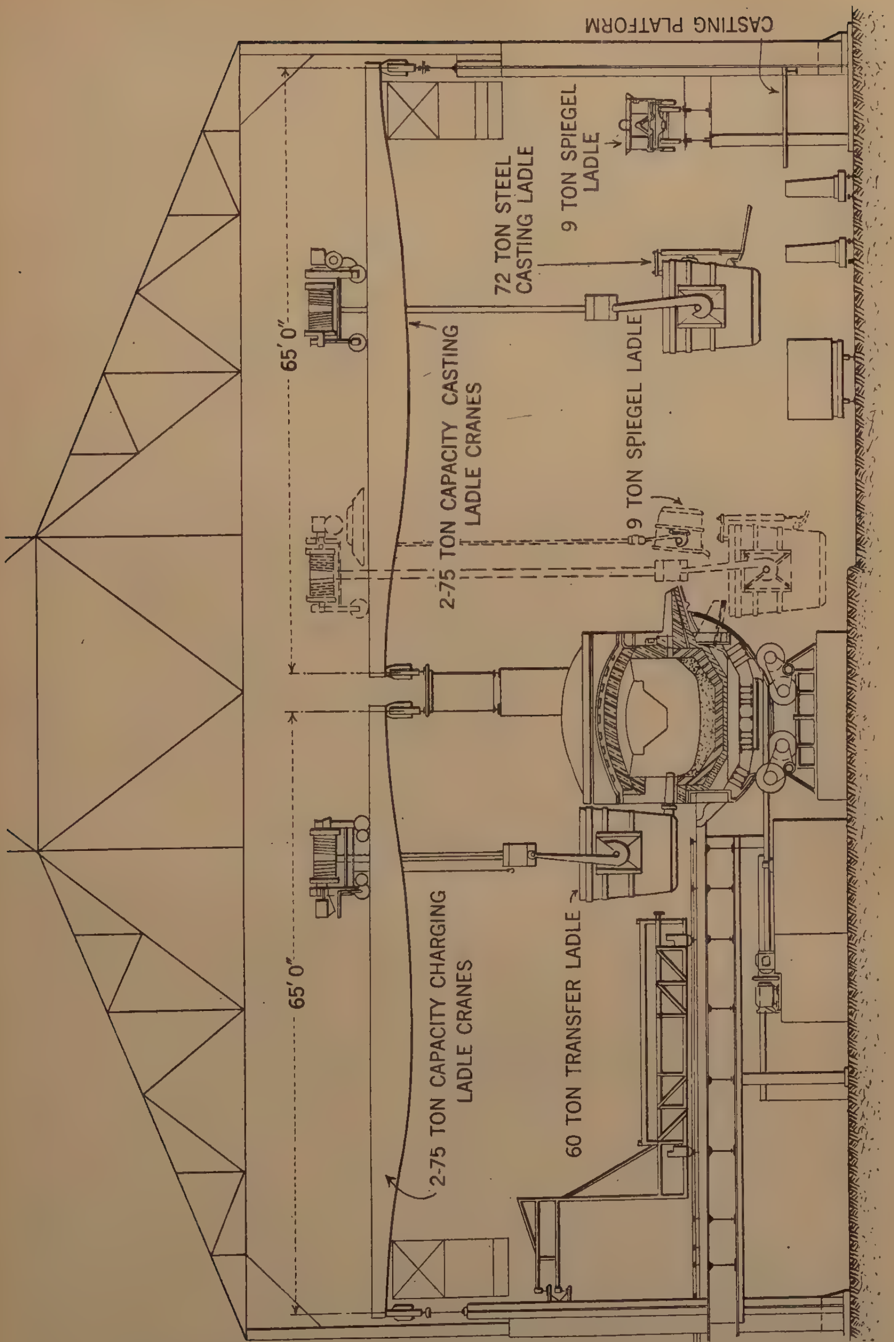

FIG. 3.—SECTION OF THE OPEN-HEARTH PLANT.

It will be noted that the dimensions of these furnaces and regenerators are small as compared to most furnaces. The ratios of the bath area and volume of the regenerators per ton rated capacity closely approach

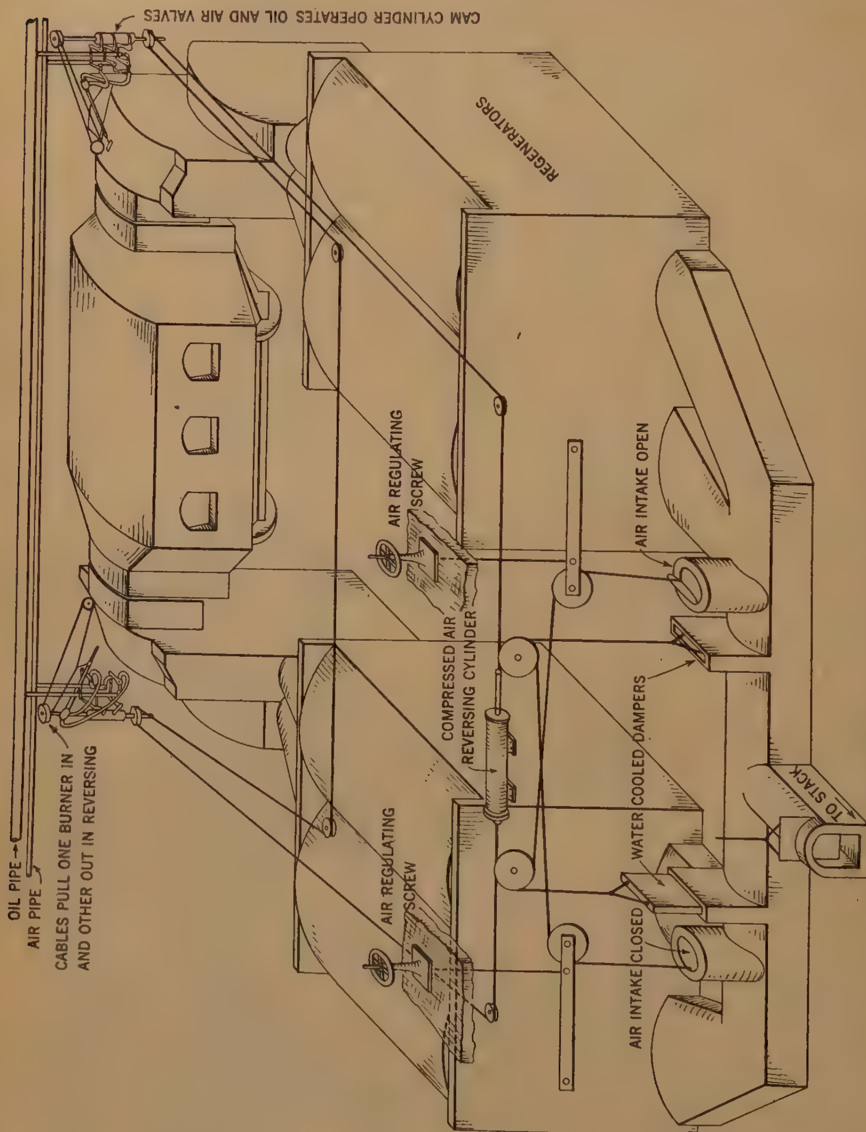

FIG. 4.—OPEN-HEARTH FURNACE REMODELED TO BURN OIL.

the continuous furnace of 200 to 300 tons capacity, in which a part of the furnace charge is tapped at one time.

In September, 1913, owing to a depression in the steel business, and the necessity of operating the plant five days per week, oil was substituted

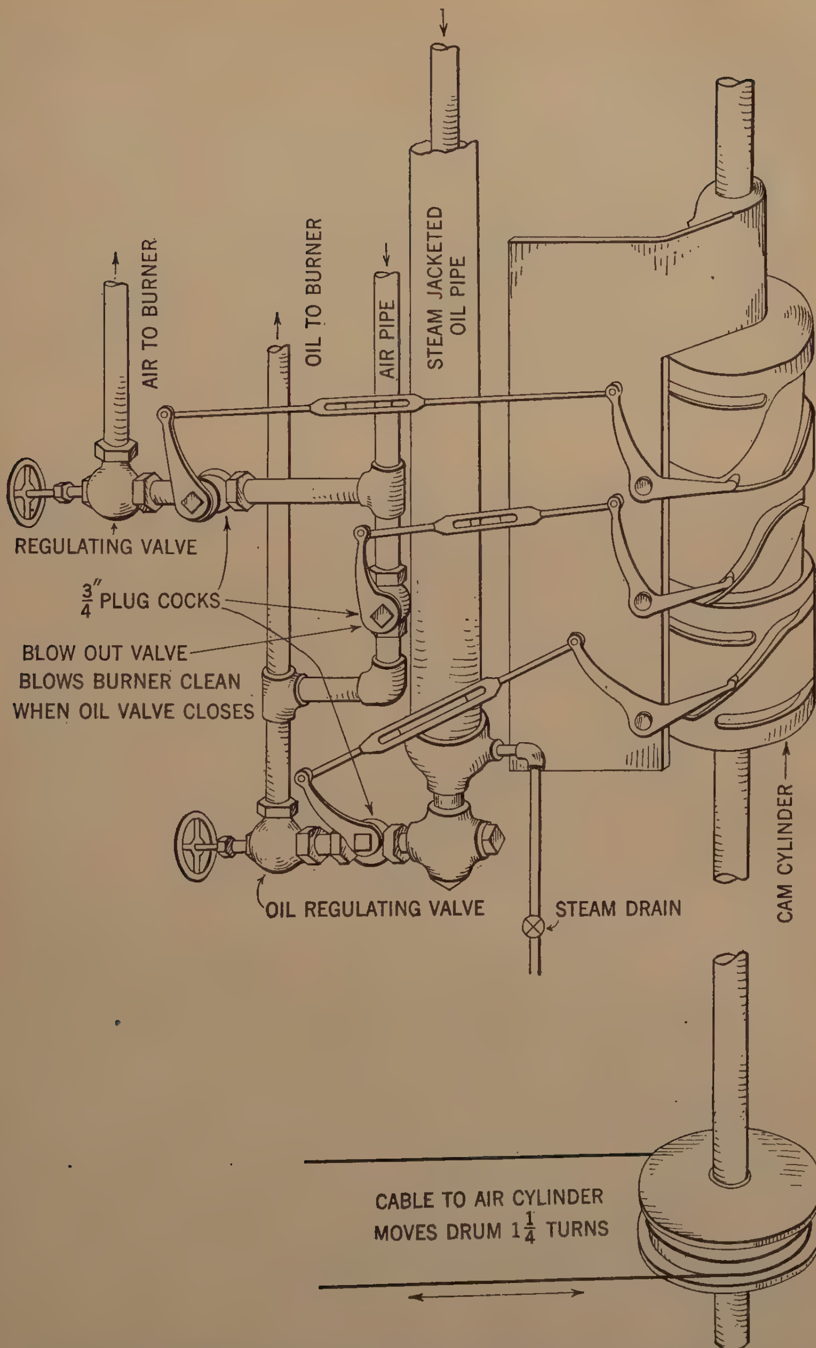

Fig. 5.—Mechanism for Controlling Operation of Oil Burners.

for gas. This did away with the necessity of carrying the gas producers over the week end, with the attendant expense. It was also found that the furnaces could be brought to the proper temperature with oil more quickly than with gas.

Fig. 4 shows a furnace remodeled to burn oil. It will be noted that there are no reversing valves, these being replaced by an ordinary damper and air lid on each end of the furnace. The waste gases pass out through one downtake to the regenerator and through one flue to the stack.

Fig. 5 shows in detail the special mechanism by which with one movement at the reversal, the burner on one end is withdrawn from the furnace and the oil shut off, and at the same time the burner on the opposite end enters and the oil is turned on.

General Operation

For one duplex heat or one complete open-hearth furnace charge, three fully blown converter charges are poured into one transfer ladle, with or without the addition of iron in the ladle for carbon. In case the iron is not added in the ladle, the necessary carbon for proper action is added in the open-hearth furnace. By this method of assembling the open-hearth charge of blown metal into one ladle in the Bessemer, the complete charging time from the Bessemer to the open-hearth furnace is brought down to about 25 min.

Converter Charge and Scrap.—A typical converter charge is made up of 75 per cent. of mixer metal and 25 per cent. of cupola metal, the latter ordinarily being a mixture of 70 per cent. of pig iron and 30 per cent. of rough steel scrap. The direct iron, containing silicon, 0.40 to 1.50 per cent.; manganese, 0.70 to 0.90; phosphorus, 0.055 to 0.065; chromium, 1.75 to 2.50; and sulphur, 0.01 to 0.10 per cent., is poured from the mixer into 45-ton ladles, and the cupola metal is added in the same ladle. This ladle makes two trips to charge the three converters. On the first trip two converters are charged, both of which start blowing at once. On the second trip, the third converter is charged. By handling the iron in this manner, a minimum skulling of the ladle is obtained. The temperature during the blow is controlled by charging from 4 to 8 per cent. of bloom crops or rail ends through scrap chutes into the converters. This, with the scrap in the cupola metal, gives a total of 14 per cent. of scrap in the charge. The high percentage of scrap added to the charge is allowed on account of the heat developed by the chromium in its oxidation during the blow. From our practical experience, it is estimated that its heat of oxidation is one-half of that of silicon.

Method of Charging the Open-Hearth Furnaces.—As soon as the converter charges are blown they are poured into the transfer ladle, which is placed on a truck and taken to the open hearth, where it is weighed,

hoisted by the charging crane, and carried to the front of the proper furnace. No runner is used in charging the metal into the furnace, each transfer ladle being fitted with an 18-in. spout to direct the stream of metal from the nozzle. The ladle is placed within 2 ft. of the furnace door and the nozzle plate is knocked off. The helper, standing on a small platform on a level with the nozzle, picks out the small clay plug with a short hook, loosens the sand in the nozzle, with which the nozzle is closed, and the stream of metal spurts out at right angles to the ladle through the furnace door. As soon as all the blown metal has run out of the nozzle, and the slag appears, the ladle is tilted back and taken to the transfer truck, where the slag contents are dumped into standard-gage steam-dumping slag bowls, and the ladle is light-weighed. The slag in the bowl

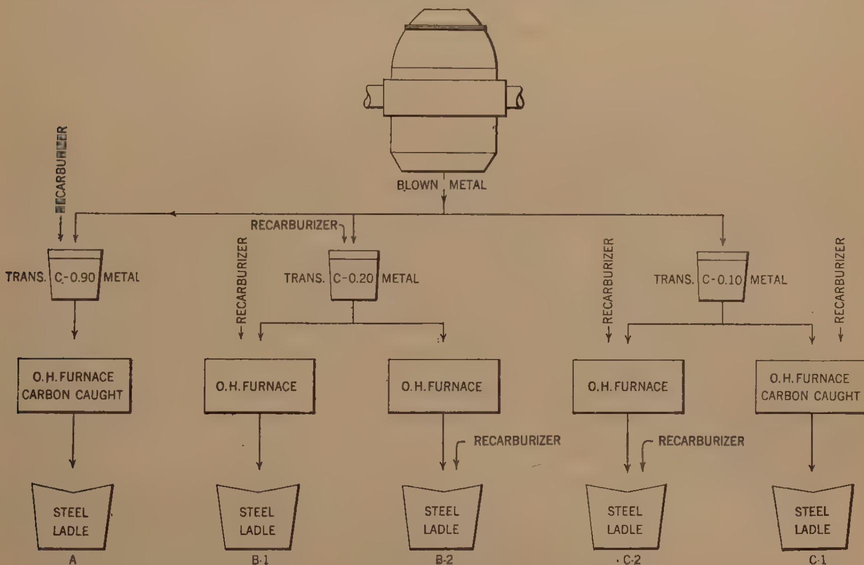

FIG. 6.—CHART SHOWING DIFFERENT POSSIBLE METHODS OF DUPLEXING.

is also weighed, and in this way the weight of the metal going into the furnace is established.

The time for running a heat from the ladle varies with the size of the nozzle, depending upon how many heats have been made through the same nozzle. It is the usual practice to take from six to eight heats off each nozzle. The total time from the pouring of the metal out of the converter nose to transferring all to the open-hearth furnace is from 22 to 28 min. There are few cases in which trouble is encountered in opening the nozzle on the transfer ladle. If trouble does occur, the metal is poured over the top of the ladle, or in case of steel in the nozzle, it is opened with a stream of oxygen. When pouring over the top of the ladle considerable acid-converter slag is carried into the furnace, which

interferes with the removal of the phosphorus and requires large lime additions for neutralizing.

Before charging, a mixture of 1 to 2 per cent. of burnt lime and 1¼ per cent. of roll scale is put on the bottom. It is desirable that the lime and scale have sufficient time to become thoroughly cut up before the duplex metal goes into the furnace. The time elapsing between tapping of one heat and charging the next varies from 20 to 60 min., an average of about 30 min. As soon as the duplex metal is in the furnace, a test is taken and sent to the laboratory, where the carbon is determined by direct combustion. The time elapsing between taking the test and obtaining the results from the laboratory is from 20 to 25 min. The treatment in the open hearth depends upon the carbon in the metal from the converters, as explained in the following description of the individual methods, outlined in the accompanying chart (Fig. 6).

Treatment in the Open Hearth

Method A.—The iron added for recarburizing the blown metal is taken direct from the mixer, and is about 20 per cent. of the total in the ladle. This method requires, with the particular iron dealt with, that the blown metal must not finish at too high a temperature, nor must the carbon be blown below 0.08, otherwise, in addition to an excessive blowing loss, there will result a violent ebullition in the transfer ladle when the recarburizing iron is added. This results in a mechanical loss due to boiling over the top of the ladle, and a loss in skulling of the ladle due to a lowering of the temperature of the steel. An advantage claimed for recarburizing in the ladle is that the addition of iron to the ladle reduces some iron from the slag back into the metal. However, a number of tests have been made on this point, and they do not show any appreciable reduction of FeO in the converter slag.

	Slag from Converter, Per Cent.	Slag in Transfer Ladle after Iron Added, Per Cent.
SiO_2	22.80	24.60
FeO	24.16	22.60
MnO	7.70	7.00
Cr_2O_3	27.65	26.10

We now have in the converter or transfer ladle 60 tons of metal of the following composition:

	Per Cent.
Carbon	0.70 to 0.90
Manganese	0.10
Phosphorus	0.065
Sulphur	0.040
Chromium	0.30 to 0.35

This metal as soon as it goes into the open-hearth furnace begins to take action. The metal loses about 10 points carbon and the phosphorus is down to 0.008 to 0.02 in the first test; the heat is, chemically, ready to tap. In the meantime, if the temperature of the bath is sufficient to go out on the first test, the melter has the tap hole open, the ladle hung, and if the analysis of the preliminary is within the limits the heat is tapped. In case the bath is not of the proper temperature, or the carbon too high, another test is already being worked by the laboratory, as the helper takes a test out of the furnace every 20 min. On receipt of the second result, if within the limits, the heat is tapped. The manganese, with this method, is generally added in the ladle, as well as the 50 per cent. ferro-silicon. If 11 per cent. ferro-silicon is used, it is added in the furnace 5 min. before tapping. A great many heats are of sufficient temperature to carry cold spiegel, added in the furnace 5 min. before tapping. Forty-five minutes is a good average time from charging metal into the furnace to tapping, and there is no doubt that this method has the advantage of rapidity and ease of operation, there being no further metal additions to the furnace, with their attendant delays and boiling effects in the furnace. The disadvantage of this method is that the time necessary to determine the carbon in the sample is of such duration that by the time the results are known the metal in the furnace has not the composition it had when the test was taken, and as the carbon does not drop uniformly, heats are tapped from time to time which do not come within the prescribed carbon limits for rail steel. This is of particular importance to this plant, as we have no other product into which heats unsuitable for rails can be rolled.

Methods B-1 and *B*-2.—These in practice were found to give unsatisfactory results, due to the metal skulling in the transfer ladle, and the fact that the small percentage of carbon often boiled out in the open-hearth furnace, before it was ready to tap, necessitating further recarburizing.

Method C-1.—The method of recarburizing in the open-hearth furnace up to 0.70 to 0.90 carbon and catching the heats "coming down" has no advantage, except that the handling capacity of the transfer ladles is increased by allowing them to be completely filled with blown metal, the recarburizing pig iron being subsequently added to the charge in the furnace.

Method C-2.—This method has been found to give more uniform carbon results in the finished steel. The iron is blown as previously described, but there is not the necessity of keeping the temperature within such close limits as when recarburizing in the converter ladle. The heats can be blown "younger"; the blown metal going into the furnace has the following composition:

	Per Cent.
Carbon	0.10 to 0.15
Manganese	0.10
Phosphorus	0.06
Sulphur	0.04
Chromium	0.20 to 0.30

The furnace, as in the other methods, has been charged with 2 to 3 per cent. of burnt lime and 1¼ per cent. of roll scale. This requires 30 min. before the duplex metal is charged. As soon as the blown metal is in the furnace, about 10 per cent. of the molten pig is added, which causes the previous dead soft metal in the furnace to take action. As soon as this iron is through the bath a test is taken, which should show approximately:

	Per Cent.
Carbon	0.15 to 0.30
Manganese	0.08
Phosphorus	0.008
Sulphur	0.05
Chromium	0.20 to 0.30

If the heat is hot enough and of the proper analysis, molten spiegel containing 20 per cent. of manganese, from the Bessemer cupolas, with enough molten iron from the open-hearth mixer to give the correct manganese-carbon addition to the heat, is poured into the casting ladle as the heat is tapped from the furnace, as shown in Fig. 3. A good average time for such heats through the furnace is 1½ hr. This, with 40 min. between heats, gives 10 heats of 62 tons, or 620 tons per furnace, in 24 hr. In following this method, the metal in the furnace must be at a higher temperature than required for the first method, on account of the cooling effect of the addition of the recarburizer in the casting ladle. This requires the furnace to be carried at a higher temperature, with a corresponding shortening of its life. The percentage of lime added is necessarily higher on account of the necessity of carrying a more basic slag to counteract the addition of the recarburizer iron in the casting ladle, with its strong tendency to reduce the phosphorus in the slag back into the metal. A typical slag contains: SiO_2, 14 per cent.; FeO, 16; MnO, 4; Cr_2O_3, 7; CaO, 45; and P_2O_5, 0.75 per cent.

Output

The maximum week's production at this plant averaged 600 tons per furnace in 24 hr. The maximum daily output per furnace has been 694 tons.

In case it should be necessary to duplex on all five furnaces to meet the increased blast-furnace output, allowed by the conditions of the market demand for steel products, the limiting factor in this plant would be the capacity of two pit cranes to handle the steel. In the final analysis of

any method or furnace, the correct factor of efficiency or speed is the tons of steel made per hour per square foot of hearth area of the furnace. From weekly averages made at different times, we find this to be 0.11 ton per hour per square foot of hearth area.

Fuel Consumption

When the furnaces were burning gas, no actual figures on coal per ton of ingots on duplex metal were available, but we are able from different runs to give some figures very close to the actual. The figures given for oil are actual meter readings.

1913	Duplexing, Including Heating Up	Cold Charge Including Heating Up
Coal per ton of ingots, pounds	235	1,030
Oil per ton of ingots, gallons	16	40
1914		
Coal per ton of ingots,	No coal used.	
Oil per ton of ingots, gallons	12.8	50

The results in 1914 were obtained under very unsatisfactory conditions, the plant operating only five days per week. Deducting the oil used in heating up the furnaces and considering the actual time the furnaces were making steel, we have an average of 9.3 gal. of oil per ton of ingots. Stated in terms of heat units per ton of ingots, we have 9.3 gal. of oil at 150,000 B.t.u. per gal., or 1,395,000 B.t.u.; assuming the available heat in the gas per pound of coal to be 10,000 B.t.u., we have $\frac{1,395,000}{10,000}$, or 139.5 lb. of coal per ton of ingots.

Life of Furnaces

On account of not duplexing exclusively on any of the furnaces, the following average for three years' operation includes about 33 per cent. of the heats made by the pig iron and scrap method: Furnace roofs, 300 heats; furnace fronts, 97 heats.

The average working time of three furnaces during this period was 5½ days per week. During this time the plant had three extended shutdowns. From observations made as to the wear of the furnaces when duplexing for different periods, we are safe in stating that the furnaces duplexing exclusively will give 500 to 600 heats to a roof, and from 150 to 200 heats to a front.

Losses

In making comparisons between the losses in the duplex process and the cold-charge process, the fact must be remembered that the difference is due to the elimination of the carbon, silicon, etc., in the converter

whereas the scrap in the ordinary open-hearth process has already withstood such a loss in its original conversion.

To satisfy the slag in Bessemer conversion 1.75 per cent. of the metallic iron is oxidized in the converter. This and the small loss in "shot" are the only losses sustained in the duplex process not common to both processes working with the same materials. The average loss from the Bessemer converter charge to open-hearth ingots for 1914 was 12.57 per cent.

With the introduction of the duplex process, there arose some criticism as to the quality of the resultant steel. As rail manufacturers, we have had a most favorable opportunity of watching the behavior in track of 250,000 tons of duplex steel and 125,000 tons of ordinary open hearth. The results of observations of these rails, of practically the same chemical composition and made under uniform conditions of rolling-mill practice, warrant the statement that there is no difference in the quality of the two products.

Measurement of the Temperature Drop in Blast-Furnace Hot-Blast Mains

BY R. J. WYSOR,* SOUTH BETHLEHEM, PA.

(New York Meeting, February, 1916)

More than two years ago, in making efficiency tests on our hot-blast stoves, I was surprised to discover a marked difference in temperature as indicated by a pyrometer inserted near a stove on blast, and the regular recording instrument situated near the bustle pipe. A special hole was then provided at the junction of the hot-blast main and the bustle pipe, and portable pyrometers with bare fire ends were used, both at this point and in the stove necks. I had occasion to make a number of such tests on various stoves on the same furnace, the difference in temperature as shown by the two pyrometers varying chiefly with the temperature of the blast, and the distance of the stove from the furnace. However, this difference was always great, often averaging above 100° F., for the entire period of a stove on blast. Later, experiments were made with similar results on another group of stoves. The pyrometers used were properly standardized and checked against each other before and after each test.

However, there was still a slight doubt as to the accuracy of the results obtained, as subsequently explained. But I have obtained corroboration of the magnitude of heat losses from blast-furnace hot-blast mains from two independent sources. I have learned from H. A. Brassert, Superintendent of Blast Furnaces, South Works, Illinois Steel Co., that he has recently done some experimental work similar to my own, and found temperature drops of the same order of magnitude. Also, C. R. Lyle, of the Armstrong Cork Co., has made some interesting theoretical calculations, based on the thermal conductivity of fire brick, for the particular conditions of several of our tests; and his theoretical interpolations, all facts being considered, check our actual observations fairly well. I understand that Mr. Lyle expects to publish the results of his investigations.

Method of Conducting the Tests

The location of the pyrometer fire ends is shown in Fig. 1, a plan of Bethlehem Steel Co.'s "G" blast furnace, hot-blast stoves, and main. Bare fire ends were used at both positions "A" and "B," being inserted

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so as to project about 8 in. inside the fire-brick lining of the horizontal stove neck and hot-blast main respectively. Readings were taken simultaneously at both positions at 3 to 5 min. intervals from the time the stove was put on until it was taken off blast. As a check on pyrometer B, we had the regular recording pyrometer located only a few feet away in the hot-blast main. The results of the latter were slightly

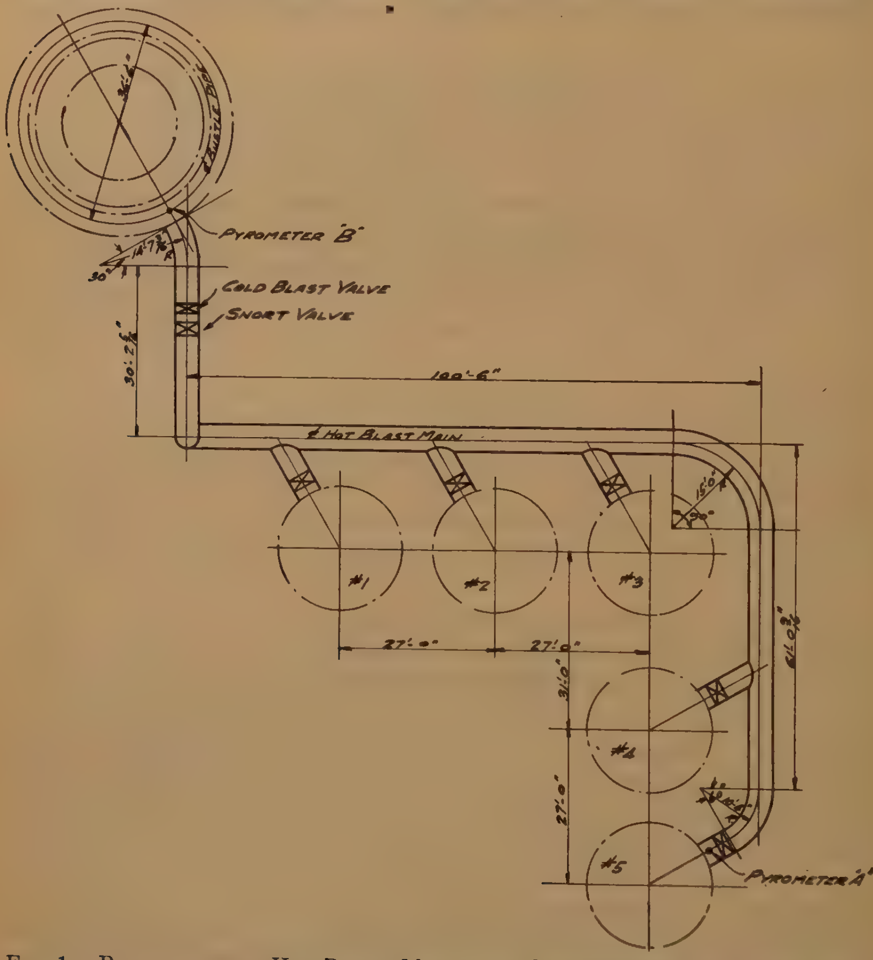

Fig. 1.—Plan showing Hot-Blast Main and Connections—"G" Furnace—and Location of Pyrometers.

lower on account of the fire end being shrouded with asbestos packing in a metal tube.

Three possible sources of error were suspected: high readings of pyrometer A, due to radiation from heated brick work of the stove neck or adjacent parts of the stove; uneven temperature of the hot blast passing through the cross-section in which pyrometer A was located;

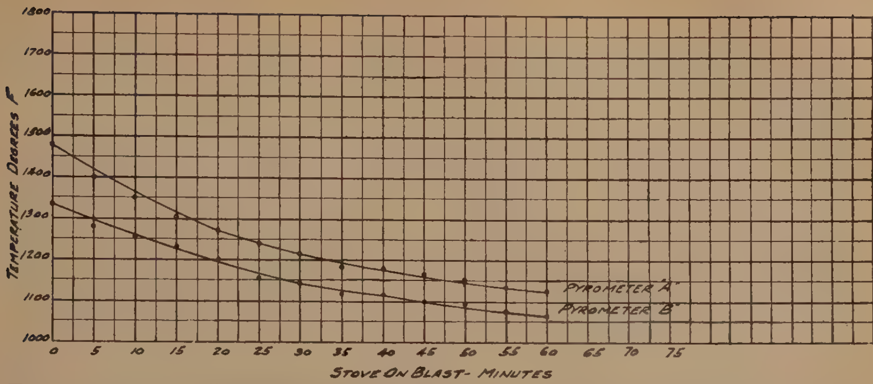

FIG. 2.—TEST 1. CURVES SHOWING DIFFERENCE IN TEMPERATURE BETWEEN POINTS IN HOT-BLAST MAIN, "E" FURNACE, WHILE STOVE WAS ON BLAST, OCT. 10, 1912.
Pyrometer A, between stove No. 5 and hot-blast valve.
Pyrometer B, at junction of main and bustle pipe.

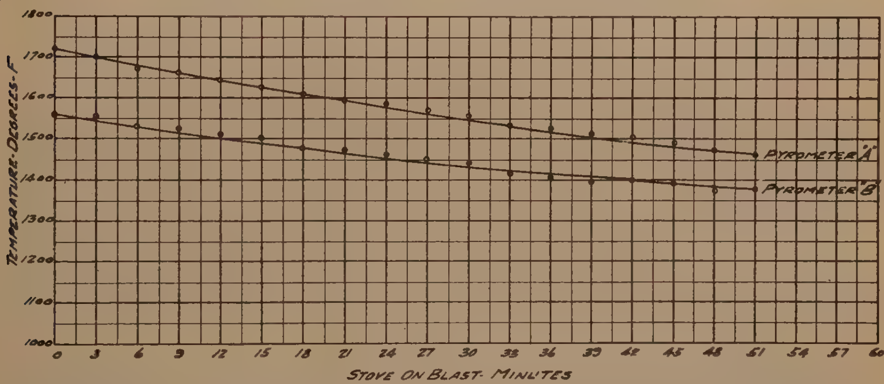

FIG. 3.—TEST 2. CURVES SHOWING DIFFERENCE IN TEMPERATURE BETWEEN POINTS IN HOT-BLAST MAIN, "G" FURNACE, WHILE STOVE WAS ON BLAST, NOV. 6, 1913.
Pyrometer A, between stove No. 5 and hot-blast valve.
Pyrometer B, at junction of main and bustle pipe.

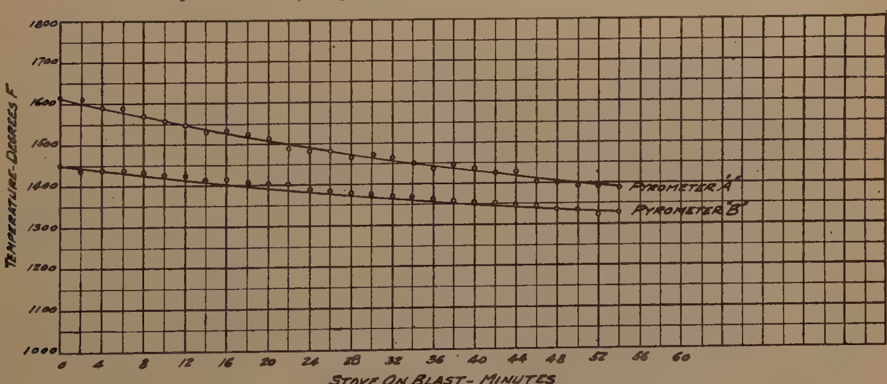

FIG. 4.—TEST 3. CURVES SHOWING DIFFERENCE IN TEMPERATURE BETWEEN POINTS IN HOT-BLAST MAIN, "G" FURNACE, WHILE STOVE WAS ON BLAST, NOV. 10, 1913.
Pyrometer A, between stove No. 5 and hot-blast valve.
Pyrometer B, at junction of main and bustle pipe.

low readings of pyrometer B due to leakage of the cold-blast valve. Both direct and indirect information leads me to believe it unlikely that appreciable error has occurred due to any of these causes.

The accompanying diagrams and tables, selected from a number, show typical conditions for the two furnaces in question. Tests were not made on other furnaces, because of the objectionable feature of breaking out brick work in the hot-blast main, with resulting danger of hot spots.

Unfortunately, no tests were made on comparatively low blast heats. On account of smelting hard, lumpy ores, we usually carry high blast temperatures at Bethlehem, and generally operate five stoves. The two runs on No. 5 stove, "G" furnace, show unusually high blast temperatures and this particular hot-blast main, for engineering reasons, is considerably longer than normal. The results show the high losses obtaining for such conditions. The "E" furnace test is more nearly representative, though the average temperature is still higher than for furnaces smelting large percentages of Mesabi ores.

General Data and Dimensions

For purposes of calculation the following data in connection with "G" furnace and hot-blast main, with connections, will be of interest. The furnace is rated at 500 tons capacity. It is served by five three-pass McClure stoves, 22 ft. diameter by 100 ft. high, connecting to the hot-blast main through Kennedy-Morrison hot-blast valves. The diameter, and the thickness of brick work in all the 500-ton furnace hot-blast mains are the same.

Outside diameter of hot-blast main	48¾	in.
Thickness of steel shell	⅜	in.
Thickness of fire-brick lining	9	in.
Length of hot-blast main on center line from hot-blast valve, No. 5 stove, to bustle pipe	192	ft.
Length of bustle pipe on center line	115	ft.
Distance between pyrometers A and B	218	ft.
Approximate total radiating surface corresponding to this distance	2,606	sq. ft.
Approximate total radiating surface of five stoves	38,500	sq. ft.
Approximate total radiating surface of five hot-blast valves and connections to stoves and main	1,286	sq. ft.
Approximate total radiating surface of hot-blast main proper	2,373	sq. ft.
Approximate total radiating surface of bustle pipe	1,474	sq. ft.
Approximate total radiating surface of 10 leg pipes	370	sq. ft.
Approximate total radiating surface of 10 blow pipes and tuyères	110	sq. ft.
Approximate total radiating surface between No. 5 stove and furnace	4,560	sq. ft.
Approximate total radiating surface between No. 1 stove and furnace	2,965	sq. ft.
Approximate total radiating surface between all stoves and furnaces	5,613	sq. ft.

For most of its length, the hot-blast main is supported 15 ft. above the ground. The stoves and entire hot-blast system are protected with black stack paint.

Logs of Three Typical Tests

The figures given under each of the subsequent tests represent actual conditions at the furnace at the time observed, except the wind velocities, which are averages for the day obtained by the Philadelphia station of the Weather Bureau; little significance is to be attached to them. The hot-blast mains of both of the furnaces concerned are fairly well protected from prevailing winds, and the actual velocity of the air passing over them probably did not exceed 4 miles per hour in any test.

Of course, the exact condition of the brick work in the hot-blast main could not be determined at the time of the tests. Only slight repairs were found to be necessary when the mains were opened after the preceding runs on the two furnaces concerned; and after being blown out again, the mains were still in fair condition, and no general repairs were necessary. Obviously we may assume that no great erosion of the brick throughout the hot-blast system had occurred up to the time of the test.

No pyrometer readings were taken for the first few minutes after the stove was changed from gas to blast.

Test No. 1

Date	Oct. 10, 1912
Furnace	"E"
Product	Standard basic
Time furnace was in blast prior to test	7 months
Time furnace was in blast after test	19 months
Stove	No. 5
Average atmospheric temperature	72° F.
Average wind velocity, miles per hour	5.8
Precipitation	None
Average blast pressure	16.3 lb.
Average free air blown per minute, at 62° F. and 30 in. barometer	32,000 cu. ft.

Pyrometer Readings

Time A. M.	Pyrometer A Degrees F.	Pyrometer B Degrees F.
10.20	1,480	1,330
10.25	1,400	1,280
10.30	1,350	1,255
10.35	1,310	1,230
10.40	1,275	1,200
10.45	1,235	1,160
10.50	1,215	1,145
10.55	1,185	1,120
11.00	1,180	1,110
11.05	1,165	1,100
11.10	1,150	1,095
11.15	1,130	1,075
11.20	1,120	1,060
Average	1,246	1,166

Total time stove was on blast	65 min.
Direct average difference in temperature between pyrometers A and B	79° F.

Test No. 2

Date	Nov. 6, 1913
Furnace	"G"
Product	Bessemer
Time furnace was in blast before test	13 months
Time furnace was in blast after test	13 months
Stove	No. 5
Average atmospheric temperature	60° F.
Average wind velocity, miles per hour	6.1
Precipitation	None
Average blast pressure	19.0 lb.
Average free air blown per minute, at 62° F. and 30 in. barometer	33,500 cu. ft.

Pyrometer Readings

Time P. M.	Pyrometer A Degrees F.	Pyrometer B Degrees F.
3.00	1,720	1,560
3.03	1,700	1,550
3.06	1,670	1,530
3.09	1,660	1,520
3.12	1,645	1,510
3.15	1,625	1,500
3.18	1,610	1,480
3.21	1,595	1,470
3.24	1,585	1,460
3.27	1,570	1,450
3.30	1,550	1,440
3.33	1,535	1,420
3.36	1,525	1,410
3.39	1,510	1,400
3.42	1,500	1,400
3.45	1,490	1,390
3.48	1,475	1,380
3.51	1,460	1,380
Average	1,579	1,459

Total time stove was on blast	55 min.
Direct average difference in temperature between pyrometers A and B.	120° F.

Test No. 3

Date	Nov. 10, 1913
Furnace	"G"
Product	Bessemer
Time furnace was in blast before test	13 months
Time furnace was in blast after test	13 months
Stove	No. 5
Average atmospheric temperature	51° F.
Average wind velocity, miles per hour	15.7
Precipitation	None
Average blast pressure	18.6 lb.
Average free air blown per minute, at 62° F. and 30 in. barometer	33,500 cu. ft.

Pyrometer Readings

Time P. M.	Pyrometer A Degrees F.	Pyrometer B Degrees F.
3.20	1,610	1,450
3.22	1,610	1,440
3.24	1,590	1,440
3.26	1,590	1,435
3.28	1,565	1,430
3.30	1,555	1,425
3.32	1,545	1,420
3.34	1,530	1,410
3.36	1,530	1,410
3.38	1,520	1,405
3.40	1,510	1,400
3.42	1,490	1,395
3.44	1,485	1,385
3.46	1,480	1,380
3.48	1,470	1,375
3.50	1,470	1,370
3.52	1,460	1,365
3.54	1,450	1,365
3.56	1,440	1,360
3.58	1,440	1,355
4.00	1,435	1,350
4.02	1,425	1,350
4.04	1,420	1,345
4.06	1,400	1,340
4.08	1,400	1,335
4.10	1,395	1,330
4.12	1,395	1,325
4.14	1,390	1,320
Average	1,486	1,382

Total time stove was on blast.. 54 min.
Direct average difference in temperature.. 104° F.

Discussion of Test Data

For sake of illustration, the approximate amount of heat lost between the farthest stove and the bustle pipe, according to Test No. 3, will be calculated.

Cubic feet free air blown per minute at 62° F. and 30 in. barometer	33,500
Pounds free air blown per minute at 62° F. and 30 in. barometer	2,548
Average temperature at stove No. 5	1,486° F.
Average temperature at bustle pipe	1,382° F.
Average difference in temperature	104° F.

Using specific-heat data on the basis of constant pressure, we find:

Total sensible heat (above 62° F.) in air blown per minute at stove No. 5, approximately	926,400 B.t.u.
Total sensible heat (above 62° F.) in air blown per minute at bustle pipe, approximately	856,400 B.t.u.
Loss in sensible heat per minute	70,000 B.t.u.
Average loss per square foot surface per minute	26.86 per cent.
Percentage loss in sensible heat above 62° F.	7.7 per cent.

If there is an appreciable pressure drop in the hot-blast main due to friction, and to heat lost by radiation and conduction, part of the above temperature and sensible-heat loss would, of course, be accounted for as due to expansion of the blast with concomitant absorption of heat. We have made no tests to ascertain the actual change of pressure between the stoves and the furnace, nor have I seen any data published on this subject. However, whatever may be the loss of temperature due directly to blast expansion, it is reasonable to suppose that expansion of the blast in turn is largely caused by heat lost by radiation and conduction into the atmosphere.

From the foregoing data it will be seen that in the tests on No. 5 stove, "G" furnace, we have measured the temperature drop due to only about three-fifths of the total radiating and conducting surface between the last stove and the furnace, along the direct path of the blast. It will be realized, of course, that there are also considerable losses between the main and the hot-blast valves of the other stoves. While the temperature will not continue to drop in the brick-lined bustle pipe and leg pipes as rapidly as in the outlying sections of hot-blast main, it will still be appreciable. And though the total radiating and conducting surface of the bare blow pipes and water-cooled tuyères is relatively small, the loss of heat per unit area is relatively great.

In the course of this investigation, I determined the average thermal loss due to cooling water from each hot-blast valve, with an average blast temperature of 1,360° F. and volume of about 33,000 cu. ft. free air per minute under standard conditions, to be approximately 10,000 B.t.u. per minute. This was done simply by measuring the discharge water from the valve and from the valve seat, and determining the temperature of affluent and effluent water. Under the conditions of the above test No. 3, this would correspond to at least 15 per cent. of the total heat loss measured in the part of the hot-blast system under investigation. The large water-cooled surfaces of the mushroom type of valve in general use, and the high thermal conductivity of these bronze or copper parts, accounts for the considerable heat loss. In this connection it is interesting to note that Mr. Brassert is installing a new type of gate valve, having much smaller water-cooled surface in contact with the blast.

The greatest heat loss in the hot-blast system occurs, of course, when the stove farthest from the furnace is on blast. However, when one of the nearer stoves is on blast, there must still be considerable radiation and conduction losses from the idle portions of the main and connections.

It is not my purpose to discuss at length in this paper the radiation and conduction losses from hot-blast stoves. A good deal has been published on this subject and most blast-furnace superintendents know about what losses are chargeable to this source.

We are now experimenting at Bethlehem with a special insulating brick for reducing heat losses from the hot-blast main. Two straight sections of the main, about 15 ft. long each, between hot-blast valve connections, have been lined with 4½ in. of special insulating brick, back of 4½ in. of fire brick, and have been in service several months. Although an appreciably lower temperature of the shell on the outside of these sections was found than on other sections of the same main, the difference was not as great as had been calculated, doubtless by reason of the high thermal conductivity of the steel shell, with consequent rapid transmission between adjacent sections. It was not deemed wise to use these bricks in bends of the main, or in the short connections to hot-blast valves or stoves. Where there is a change of direction, fire brick are eroded more rapidly by the "sand-blast" action of dust in the hot blast; and if the inner course of fire brick should be pierced, the soft insulating brick would be cut out with comparative rapidity. This doubtless would be likewise the fate of any other ordinary insulating material similarly used.

Estimates have also been prepared for external insulation with 1½ in. pipe-covering blocks, a 1-in. air space being provided. Besides the possible objections to appearance, first cost, and maintenance, the question as to possible overheating of the shell, with consequent warping or unequal expansion, must be considered.

In the construction of a new hot-blast main, the logical plan would apparently be to design it of sufficient diameter to permit lining with 9 in. of fire brick and a backing of several inches of insulating material.

Conclusions

1. There is, between the stoves and the blast furnace, a very marked drop in the temperature of the hot blast, which is not only serious as to temperature but signifies also a thermal loss. It is affected by many variables, such as size and length of hot-blast mains and connections, thickness of fire-brick linings, volume and initial temperature of air passing per unit of time, temperature and wind velocity of the outside air, extent of water cooling, etc. It is probable that, at a great many furnace plants in the country, these losses are of sufficient magnitude to

justify an attempt at their reduction. The present tendency is toward higher blast heats on furnaces carrying large percentages of Mesabi ores in their mixtures; and high heats have always been desirable in smelting hard, coarse ores, such as the magnetites so generally used in the East. In the manufacture of spiegel and ferro-manganese, the maintenance of high blast temperatures is of particular importance. In all plants where economy in the use of blast-furnace gas must be practiced, on account of its large consumption in gas engines or under boilers, this is likewise a matter of special interest.

2. That insufficient insulation in the hot-blast main is the cause of serious temperature and heat losses, will be readily recognized. It seems logical and practicable to prevent a large percentage of these losses by means of one of the methods of insulation suggested in this paper. With such insulation, other operating conditions being normal, we would expect higher available blast heats with resulting fuel economy, or a smaller quantity of fuel gas necessary for heating the stoves I believe that blast-furnace managers and superintendents generally will be interested in a study of better insulation of the hot-blast main and stoves.

In conclusion, I wish to acknowledge the practical co-operation of one of my assistants, H. A. Floyd, in securing the data used in this paper.

Discussion

Leonard Waldo, New York, N. Y.—Will Mr. Wysor kindly give us more information as to the pyrometers used and the temperature measurement?

R. J. Wysor, So. Bethlehem, Pa.—We used standard Bristol pyrometers with bare fire ends. These were standardized both before and after the tests with salt and pure aluminum. We also interchanged the pyrometers.

J. W. Richards, So. Bethlehem, Pa.—Did you attain any results which gave you approximately the value of the heat insulation at any of the furnaces?

R. J. Wysor.—In the first installation we were afraid to put in the insulation throughout the hot-blast main for the reasons mentioned in the paper, but we used the insulating brick in two 8-ft. sections in straight lengths of our hot-blast mains. Owing to the high conductivity of the steel shell, the temperatures which were taken on the outside were not representative of the two temperatures resulting from heat flow through the insulating brick. I believe that when a hot-blast main is insulated so as to cut off the heat flow along the steel shell, the actually determined

external temperatures will be comparable with theoretically calculated results.

J. W. Richards.—I have been talking in season and out of season about painting furnaces white. A technologist of a very large plant told me last week that he started to paint the furnaces white, in his plant, in order to save heat; at first he had difficulty in getting the men to do it, until they found that it made the furnace room so much more comfortable that they were glad to keep the furnaces white by whitewashing. My personal conviction is that a steel company would be well repaid in the amount of heat saved, by continually painting its furnaces white, using perhaps aluminum paint where whitewash is not suitable.

Linn Bradley, New York, N. Y.—In connection with measuring the temperature of the gases, the author speaks of the pyrometric couple being exposed to the radiant energy from the heat stored in the brick work. The same error would be caused by the dust particles passing through with the gases. Mr. Wysor has pointed out very clearly in his paper how it is possible to regulate this matter.

C. R. Lyle, Pittsburgh,* Pa. (communication to the Secretary†).—The tendency in the last few years seems to be toward a fuller realization and appreciation of the fact that aside from tonnage other items play a very important part in the cost of blast-furnace output.

In connection with other work in the insulation field, the enormous waste, due to improper insulation of stoves and blast mains, suggested itself, and some calculations were made with the idea of arriving at an approximate idea of the probable losses. These calculations were brought about at the suggestion of R. J. Wysor, Chief Chemist of the Bethlehem Steel Co., and are referred to in his paper.

I shall present some calculations showing the losses that might be expected from stoves and mains, also the temperature drop in the mains, and at least one important step in the direction of reducing these losses.

As just pointed out, there are two phases of loss or decreased output occasioned by the heat loss from the hot blast main that deserve consideration. One is the well-known radiation loss, which can be measured in B.t.u., and of course is a factor in the fuel cost. The other phase is not quite so tangible a loss, and yet in the minds of most steel men with whom it has been discussed it seems to be considered the more important. This is the handicap imposed by the temperature drop between the stoves and the furnace, due to radiation loss through the mains.

You will note from Mr. Wysor's paper that this loss in some cases amounted to 190° F., the average, of course, being considerably less. In

* Ass't Mgr., Armstrong Cork & Insulation Co.

† Received Feb. 18, 1916.

the stoves, the problem is largely one of fuel economy, although there is the element of increased temperature to be gained by reducing the heat losses. In the discussion of these losses I have confined myself, as far as possible, to the conditions surrounding Test No. 3 of Mr. Wysor's paper, which was made on Nov. 10, 1913, on G furnace. The temperature in the third pass of the McClure stove was taken from data given me from determinations made in plants in the Pittsburgh district, due allowance being made, of course, for the difference in blast temperatures employed in the two cases.

In calculating losses from the present construction in both stoves and mains, it is, of course, necessary to know the transmission through firebrick, as that is the lining used. In consulting the publications on this subject, I was rather surprised to find a wide variation in the results of the different authorities. In fact, this was so great that, in this discussion, the average of the results given by three authorities for temperatures ranging close to 1,400° F. was used. Those taken in reaching this average were the results of Dougill, Hodsman & Cobb, Wologdine and Dudley, and the value arrived at was 258 B.t.u. per square foot, per degree difference in temperature, per 1-in. thickness, in 24 hr. To make the losses more apparent, I have decided to show losses with the present construction, consisting of 9-in. firebrick lining; also in parallel with that, the losses when the insulating medium is used in the construction. The insulating medium selected is one of which I have intimate knowledge, a brick composed of cork and diatomaceous earth with a clay binder. In the process of manufacture the cork is burned out, leaving a light, porous, highly efficient insulator. Tests which have come directly under my supervision and have been checked by at least two disinterested engineers, show the transmission through this material, at a temperature of 1,400° F., to be 21 B.t.u. per square foot, per degree difference, per 1-in. thickness, per 24 hr.

It will probably best serve our purpose to discuss first the temperature drop in the hot-blast main. Before doing this it is necessary to establish the conditions and units used in this calculation. The existing hot-blast main is lined with 9 in. of firebrick. It will, therefore, probably be a more striking illustration if we base our calculations on replacing one layer of firebrick with one layer of insulating brick. In other words, one set of figures are based on the 9-in. firebrick lining, as at present installed, and the other on 4½-in. firebrick backed up by 4½-in. insulating brick.

From the results of Test No. 3, the average temperature of the hot-blast main is 1,434° F.; the average air temperature outside—51° F.; air blown per minute at 62° F. and 30-in. barometer—33,500 cu. ft.; approximate equivalent air at 1,434° F. and 18.6 lb. pressure—53,790 cu. ft.; temperature difference between air in main and air outside—1,383° F.; transmission through firebrick, per 1-in. thickness, per degree difference,

per 24 hr.—258 B.t.u.; transmission through insulating brick, per 1-in. thickness, per degree difference, per 24 hr.—21 B.t.u.

This gives us a transmission through 9 in. of firebrick of 1.195 B.t.u., per degree difference, per hour, and a transmission through the insulating brick and firebrick combination of 0.18 B.t.u.

The length of hot-blast main between pyrometers A and B is 218 ft., the outside diameter being 49¾ in.

The area of the blast main at the mean circumference of the brickwork is 2,225.78 sq. ft.

The heat loss from the blast main with a 9-in. firebrick lining is, of course, the product of the loss per square foot, per degree temperature difference, times the temperature difference, times the number of square feet; or, in other words, 2,225.78 by 1,383 by 1.195, which equals 3,678,513 B.t.u. per hour.

Similarly, the loss through insulating brick and firebrick combination is 2,225.78 by 1,383 by 0.18 = 554,085 B.t.u. per hour. This heat loss, of course, acts directly to reduce the temperature of the air in passing through the blast main.

We are pumping 53,790 cu. ft. of air per minute, and the area of the main is 4.9 sq. ft., which gives us a velocity of 10,977 ft. per minute. This means that it would take the air 218 ÷ 10,977 = 0.0198 min. to pass through the main, which is equivalent to saying that each particle of air is acted upon by the heat loss from that main for a period of 0.0198 min., or the total volume of air in the main at any one time is acted upon by the total heat loss from that main for a period of 0.0198 min. The amount of air passing through the main in 0.0198 min. is 1,065 cu. ft., which weighs at this temperature and pressure, 0.04733 lb. per cubic foot, or 50.4 lb. Taking the specific heat at constant pressure, we find that 11.97 B.t.u. is required to raise or lower the temperature of this air one degree.

We found that the total loss per hour through the main with the firebrick lining was 3,678,513 B.t.u.; with the insulating brick and firebrick combination lining 554,085 B.t.u. per hour. This is equivalent to 1,213.9 B.t.u. in 0.0198 min. through the firebrick and 182.85 B.t.u. through the combination firebrick and insulating brick. Therefore, if 11.97 B.t.u. is required to lower the temperature of this air 1°,

1,213.9 ÷ 11.97 = 100.3° F. temperature drop in the hot-blast main with 9-in. firebrick lining,

182.85 ÷ 11.97 = 15.3° F. temperature drop between pyrometers A and B with 4½-in. firebrick and 4½-in. insulating brick combination lining in the main,

showing that the temperature drop may be reduced 85° if the combination firebrick and insulating brick lining is used.

It is also rather interesting to note that the theoretical temperature drop, as determined, is not far from the average shown by Mr. Wysor in his Test No. 3, and this, of course, is to be expected, as the temperature conditions assumed in this calculation were the averages of his test.

We will now take up the question of heat loss, both in the stoves and main. We found that the heat loss through that portion of the main between pyrometers A and B was 3,678,513 B.t.u. per hour through the firebrick lining, and 554,085 B.t.u. with the combination insulating brick and firebrick lining. To this must be added the heat loss from bustle pipe and leg pipes, etc., the area of which is 2,897 sq. ft. The loss from this surface, with the firebrick lining, is 4,787,582 B.t.u. per hour; with the combination firebrick and insulating brick lining 721,179 B.t.u. per hour. This gives a total loss between all stoves and the furnace of 8,466,095 B.t.u. per hour for firebrick and 1,275,264 B.t.u. per hour for the combination insulating brick and firebrick.

It is now necessary to determine the stove losses. The stove in question is a three-pass McClure stove, which is assumed to have a lining composed of 9 in. of firebrick between the third pass and the metal, and the comparison is made between this lining and the combination 4½-in. firebrick backed up by 4½-in. insulating brick. It is difficult to find data from which to compute the average temperature difference between the air in the third pass and the outside air, one reason being that this varies so greatly at different periods of operation. The bottom, from the information given, varies between 900° to 500° F. The temperature of the top, of course, varies from the incoming air temperature to probably 600° or 700° F. The stove is on gas four-fifths of the time and on air one-fifth, and it would seem that the fair average temperature difference for the entire pass is 490° F.

The total radiation surface of the five stoves, therefore, would be 38,500 sq. ft. The total heat loss from this area, with 9 in. of firebrick and 490° temperature difference, is 22,543,675 B.t.u. per hour. The total heat loss, on the other hand, with the combination 4½-in. firebrick and 4½-in. insulating brick lining, is 3,395,700 B.t.u. per hour.

This loss, added to the loss from the mains and connections, gives a total loss for the firebrick lining of 31,009,770 B.t.u. per hour.

The total loss through the insulating brick and firebrick combination is 4,670,964 B.t.u. per hour, or, in other words, the saving by the use of insulating brick is 26,338,806 B.t.u. per hour, which is equivalent to 230,727,940,560 B.t.u. per year.

Taking gas at a thermal value of 110 B.t.u., this is equivalent to saving 239,444 cu. ft. per hour, or 2,097,526,732 cu. ft. per year. Based on the assumption that these stoves require 17,000 cu. ft. of gas per minute, this would mean a saving of approximately 23.5 per cent. The saving on the stoves alone is 17.06 per cent.

No doubt this saving will vary considerably, and it is probably larger than would be shown in a great many plants. It is necessary to take into account, in considering this case, the fact that we have a particularly long main with high temperature blast, and in addition to this the areas taken for all, except that portion between pyrometers A and B, are outside surface of pipe instead of mean area of brickwork. It is impossible to determine from the figures at hand what the correct area at mean circumference should be. It should be evident to all of us, however, that these losses are sufficiently large to deserve careful consideration, and earnest effort should be made to reduce them to a minimum.

L. A. Touzalin,† South Chicago, Ill. (communication to the Secretary).—With the present tendency toward higher blast heats, together with the increased importance of greater economy in the use of blast-furnace gas, the conservation of heat throughout the entire hot-blast system is becoming more and more important.

Realizing that any improvement which would tend to better conserve the heat of the blast would undoubtedly result in greater benefit to our furnace practice than the same effort expended along many other lines, we have been experimenting for some time on several methods of cutting down the loss discussed by Mr. Wysor in his very interesting paper. Our experiments were not only concerned with the heat lost by radiation, due to insufficient insulation, but also with the loss of heat due to the cooling water used along the hot-blast system. It is unnecessary to discuss at much length the loss of heat by radiation, as Mr. Wysor has gone into considerable detail in showing its amount and how it can best be minimized. Suffice to say that we are now installing a special insulating lining between the steel shell and the brickwork in our stoves, as well as hot-blast main and bustle pipe.

In regard to the heat loss due to cooling water, it may be of interest to mention that we have derived certain advantages from the installation of a gatetype of hot-blast valve in place of the more common mushroom type. In the first place the gate-type valve was found to possess several mechanical advantages over the mushroom type, of which the following are the more important: When seated, this valve is practically leak-proof, due to the fact that the pressure of the blast on one side of the gate holds the gate firmly against a flat seat of larger area than the tapered seat of the mushroom type. The construction of the hot-blast main is much simplified with this type of valve by avoiding the right-angle bends required for the mushroom type. In opening and closing the valve it is positive in its action and easy to operate. The characteristic of this valve, however, which is especially noteworthy in connection with a discussion of heat losses, is the fact that a much smaller water-cooled surface

† In Blast-Furnace Dept., South Works, Illinois Steel Co.,

is exposed to the hot blast. We have found that this results in a certain saving of heat, which is best shown by means of the curves shown below.

These curves are the graphical results of a series of tests made on both types of valves. The temperature of the entering and discharged cooling water was observed, and the water measured, at regular intervals. The average blast volume was about 40,000 cu. ft. free air per minute, as

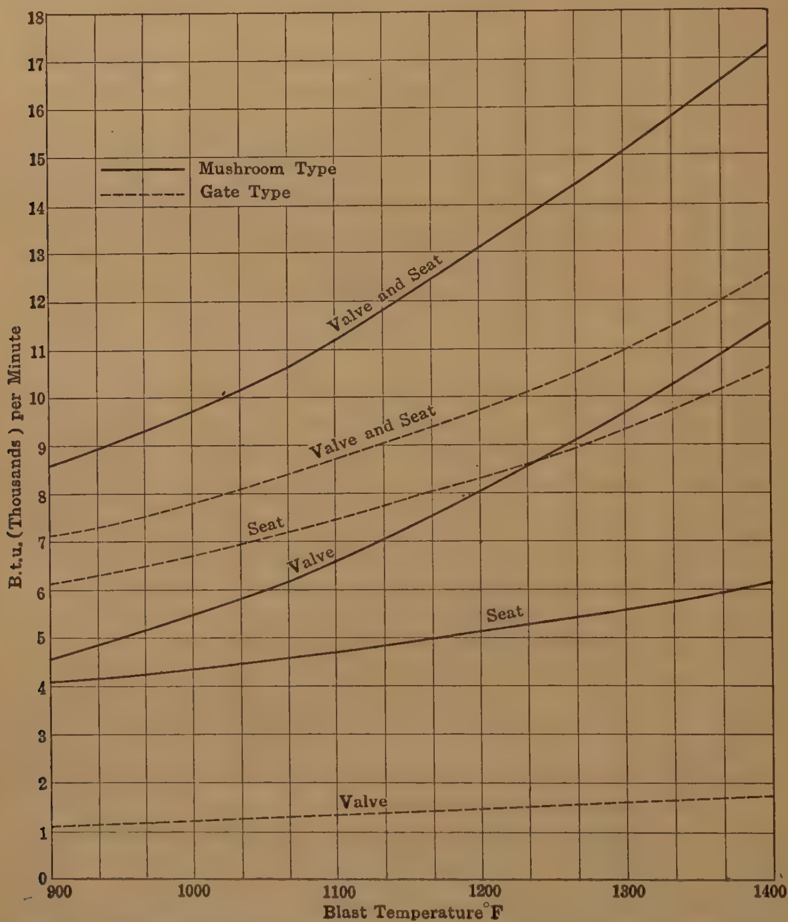

FIG. 1.—CURVES SHOWING SAVING OF HEAT WITH GATE-TYPE VALVES.

measured by engine revolutions. The temperature of the blast was obtained by inserting a thermocouple, protected by a steel tube, into the hot-blast main at the center of the hot-blast valve, when the gate valve was used. When testing the mushroom-type valve the experiment was conducted on a stove of a different furnace but situated almost exactly the same distance from the bustle pipe as was the gate-type valve from

its bustle pipe. The total distance between the pyrometer at the gate valve and the pyrometer at the bustle pipe was about 40 ft. This distance is much less than that shown on p. 374. For this reason the decrease in temperature was not found to be as great as shown by Mr. Wysor; our pyrometers registering within 15° to 20° of each other.

Inasmuch as the total heat loss is the sum of loss by radiation and loss due to cooling water, it would seem proper to show these losses separately. Mr. Wysor gives a figure for the average heat loss per square foot of surface per minute. If the loss due to cooling water were subtracted from the total loss, the balance, divided by the square feet of surface, would give a more accurate approximation of the heat loss due to insufficient insulation.

P. A. Boeck, New York, N. Y. (communication to the Secretary*).—This very interesting and valuable paper is of great importance in giving quantitative data on a subject regarding which authentic information is very difficult to obtain. It is of great importance not only to the iron and steel industry, but also to other lines of work in which high temperatures are employed.

In considering the insulation of high-temperature equipment, I believe that the most important factors are not those which have received the most attention (usually the saving in fuel incident to proper insulation), but rather the effect in increasing the capacity of the equipment, the control of the temperature, and the uniformity of heat throughout its various parts.

The fact that a furnace is not productive until a certain temperature is reached emphasizes the point that the heat available above the required temperature at which the reaction begins to take place is of far greater value than any quantity of heat below this effective temperature. In other words, heat is being added at the point of maximum effectiveness, and is gaged by the increased production of the equipment. It is, of course, necessary to consider the nature of the reaction and the most effective temperature at which the reaction is effected, as well as the effect upon the refractories. This point was brought out strongly in a discussion of a recent paper on high-temperature insulation before the Chicago Section of the American Society of Mechanical Engineers.

To those informed on iron and steel works practice, it is an established fact that often for a period of varying extent each winter, the furnace production is suspended during cold snaps, because it is impossible to obtain the requisite temperature and distribution of heat to proceed with the operation. This is especially noticeable in blast-furnace practice, where this equipment is subject to the full effect of atmospheric conditions. This decrease in production could undoubtedly be largely reduced

* Received Mar. 3, 1916.

or entirely prevented by the proper precautions being taken in the matter of insulation.

The data given by Mr. Wysor has been partially verified at other plants in which insulation has been installed. Complete data are, however, not at the present time in the proper condition to be discussed in detail.

It is hoped that further detailed information of this character may be obtained from other industries, for the question of heat insulation is, I believe, one of the most important factors in the economical operation of high-temperature equipment.

R. J. Wysor.—In conclusion I may state that within the last few months insulation of one form or another has been installed in or planned for the hot-blast system or stoves, or both, at a number of blast furnaces in various sections of the country.

Washed Metal

BY HENRY D. HIBBARD,* B. S., PLAINFIELD, N. J.

(New York Meeting, February, 1916)

The process of making washed metal now followed is in principle that described by Holley before this Institute in 1879, as the Krupp washing process. Sir Lowthian Bell also experimented extensively in England during 1877 and 1878 on refining iron and joined his interest with Krupp, the process being often referred to as the Bell-Krupp process. The only plant in this country now operating this process is that of the Brier Hill Steel Co., Youngstown, Ohio. This process may be considered as the sole survivor of the numerous methods conceived for refining crude iron, when the removal of phosphorus from steel-making material was the great desideratum affecting large districts. The east coast of England and all northern Germany, the homes of Bell and Krupp, respectively, could at that time make steel only from imported ores. These processes were in use or in various stages of development 40 years ago and all except the Bell-Krupp received their quietus from the development of the basic steel processes, both pneumatic and open hearth. The aim of the early refining processes was, first, the removal of the silicon, which could be done on a sand bottom, and later, the removal of the phosphorus, requiring a bottom which would give or permit a basic slag. The incidental advantages of increased yield and small fuel consumption were not generally recognized or considered; the aim of the later operations was the saving of the difference in value between low and high phosphorus irons, as well as hastening the removal of the metalloids.

The chemistry of the washing process follows in part that of the puddling process, and indeed washing might be considered as the first half of puddling, since all the non-ferrous elements of the crude iron, except the carbon, are eliminated and the iron is maintained molten until cast. In puddling, the iron loses its carbon as well, and because of that, its fusion point rises above the hearth temperature; it freezes into a pasty malleable form, and is worked at once by squeezer, hammer, or rolls.

Since Holley's paper, important improvements have been made from time to time in both plants and methods for washing iron, and the

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* Consulting Engineer.

degree of purification attained is far greater than Krupp accomplished, particularly as to phosphorus, though Bell's results approximated those of the present practice. The costs have been diminished as well.

The use of washed metal is limited. It is in request now only as an exceedingly pure iron and is used, first, in acid steel processes, solely for the reason of its purity, as in open-hearth and Bessemer steels of high grade; and, second, because of its high carbon contents (3.25 to 3.50 per cent.) as well as purity, as a carburizer in crucible steel.

Washing Plant of Brier Hill Steel Co.

The washing plant of the Brier Hill Steel Co. (Fig. 1) is placed near one of its blast furnaces, which supplies it with molten crude iron. It consists briefly of a reservoir, a Pernot furnace, a reheating furnace, and a casting machine.

The Reservoir.—The reservoir is about 100 ft. or more from the blast furnace, a runner in the floor of the cast house conducting the stream of molten crude iron to it from the furnace. It has a brick lining 18 in. thick, is of 50 tons capacity, and is mounted on top of the ram of a vertical hydraulic cylinder, which straddles a 100-ton weighing scales. It is provided with mechanism for tilting to pour the metal. Fifty tons of iron make four washed-metal charges, which take about 1 hr. each to work, and, since the blast furnace is tapped every 4 hr., the reservoir provides a continuous supply of iron to the washing plant. The reservoir receives iron when at its lowest position, is raised by the hydraulic plunger about 15 ft., and discharges to a short open runner which conducts the molten crude iron to the Pernot furnace. Iron has been held molten in the reservoir 48 hr. with the help of a small oil jet, and then washed without difficulty.

The Washing Furnace.—The Pernot furnace in which the washing is done (Fig. 2) differs in many details from the Pernot furnaces which have been described. In fact, nothing of the original furnace remains and hardly any of the details of the original design, though the general plan is the same. The changes made have resulted in greater durability and efficiency, but even now, the cost of repairs is an important item in spite of the relatively low temperature of the operation.

The pan, as the circular hearth is termed, was originally designed to revolve on wheels, but now runs on conical rollers, which are kept spaced by a spider. The rollers travel on a circular track below the pan, turning in the same direction as the hearth, but at half the speed. On the bottom of the pan is a circular track on which it travels over the rollers. The pan is driven by a bevel pinion engaging a gear on the bottom of the pan. The pan lining is so thin (8 in.) that water is continuously sprayed upon the bottom, from underneath, to keep it cool.

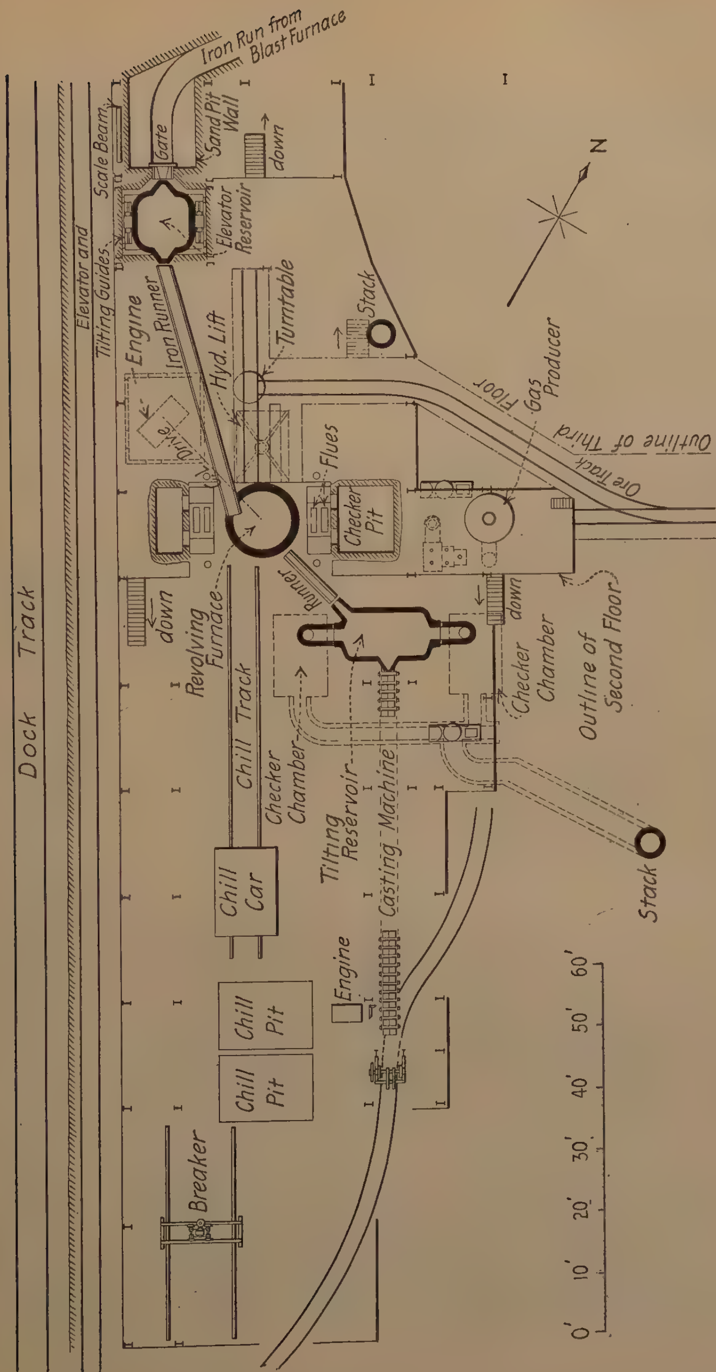

Fig. 1.—Washed Metal Plant of Brier Hill Steel Co.

In the superstructure, at one side of the single operating door on the charging side and above the pan, is an opening with which the crude-iron

FIG. 2.—FRONT VIEW OF WASHING FURNACE, SHOWING TIPPING RECEIVER FOR HOLDING WASHED METAL WHILE CASTING IN MACHINE ON THE RIGHT.

runner from the reservoir is connected and through which the molten iron is charged into the furnace. In the side of the pan near the bottom

is a tap hole and near the top a slag notch. A hole through the furnace roof connects with an ore chute through which the ore is shot for fettling the hearth after each charge has been drawn off.

One circular gas producer supplies gas for the use of the furnace. The ability of a single producer to run this plant, depends on the fact that the coal used contains only about 1.5 per cent. ash, with 44 to 46 per cent. of volatile matter, and will not coke under any circumstances. This makes it possible to carry a fire 8 ft. or more deep, and drive at a high rate with a steam injector for the air, without having above 2 per cent. CO_2 in the gas. This coal in earlier days was used raw in the blast furnaces of the neighborhood without other fuel for making pig iron and is in this respect like the Scotch coals.

A section of the floor, 10 by 10 ft., on the charging side of the furnace, is arranged to be lowered, which is done after each heat is run out, providing a place for the men to stand to repair the tap hole and cinder notch, the pan being revolved so that these openings are on the high side.

Operation.—After a heat has been run out, the furnace is drained; any holes in the hearth are puddled out and filled with iron ore, and the fettling, consisting of dampened ore, is run down in successive batches through the roof, the hearth being revolved a little at each dose, so that the bank of fettling is continuous around the hearth. The gate of the ore chute above is controlled by the melter from the working platform by means of overhead levers. While the banks are thus being built up, the bottom is covered with ore thrown by the shovelful through the operating door. The ore used for bottom making and fettling is a high-grade specular hematite, broken into pieces from the size of a hen's egg down, with enough fine ore to make a compact mass without voids.

Between 20 and 30 min. are consumed in repairing the bottom and fettling. The tap hole and cinder notch are cleaned out and plugged with sand.

The firing is then resumed to heat and set the fix, the reservoir is raised, and 12 tons of iron is run into the pan, which is then started revolving. A reaction immediately takes place between the oxygen of the ore and the silicon, phosphorus, and manganese of the iron, the bulk of these elements being eliminated in a few minutes, by passing into the slag which is formed. The small percentage of carbon which goes off during the operation (3 to 5 per cent.) makes enough CO gas to give a vigorous boiling action, throwing jets of slag 3 to 5 in. above the bath at first when the temperature of the bath is greatest.

About 15 min. after the crude iron has been charged the slag is made to run off through the slag notch. This slag contains most of the oxidized silicon, phosphorus, and manganese from the crude iron, and perhaps a quarter of the sulphur. A second slag is then formed by additions of lime, which, with the ore melted from the fettling, takes up more of the

phosphorus and sulphur of the iron. These slags are rich in iron and are therefore smelted afterward in the blast furnaces for making basic pig.

Tests are taken from time to time, averaging about five to a heat, to determine the degree of dephosphorization. The test is a circular cake about 5 in. in diameter and 1.25 in. thick in the center, being thinner at the edges. This cake is broken into pieces. The test contains a myriad of small bubbles and has a faint crystalline or columnar structure normal to the bottom surface. The lower the phosphorus, the larger the bubbles, and the fainter the crystalline structure mentioned. With very low phosphorus, bubbles as large as 1/16 in. in diameter may be found within 1/4 in. of the lower surface, with smaller ones still lower and larger ones up to 1/4 in. in diameter near the upper surface.

The whole washing operation takes about an hour, the most of this time being consumed in securing the extremely low phosphorus content required. When ready for casting, the tap hole is turned to the proper position on the pit side of the furnace and is then dug open by means of a hand bar.

The metal runs from the furnace fairly well, but looks cool, almost of an orange temperature, perhaps not over 1,250° C. It gives off copious brown fumes, as well as scintillating sparks and CO gas, which burns, the whole making a brilliant effect and, when viewed from a distance, gives one the impression that the metal is hotter than it really is.

Casting.—For casting the metal two means are provided: The one generally used when the product is for domestic consumption is a broad shallow cast-iron pan on a car, which runs on a track leading from the Pernot furnace down the center of the cast house. The charge is run out into this pan, where it forms a single plate from 4 to 6 in. thick, and the car is then drawn away from the furnace to cool. Water from a hose is played upon this plate for a few minutes, while hot, to loosen any slag which may come out after the metal and is not trapped off while casting. When the cake of metal is cool enough to be moved without breaking, but is still red hot, it is lifted from the car with a crane and put in a pit. Water is then turned on and it is flooded until cold enough to handle, when it is broken up into one-man size chunks, either with a sledge or by a traveling breaker. This is not difficult, because the water cooling has filled it with cracks, which go clear through the plate. These cracks seem to have no system, except that they are always at right angles to the upper and lower surfaces of the plates, leaving the pieces very irregular in outline.

The other casting method, usually employed for export orders, is to run the washed metal without the slag, into a 12-ton tilting, oil-fired regenerative furnace, located at a lower level, where it is held about 15 min., heated, and poured directly into the molds of an endless-chain casting machine, which moves at the rate of about 15 ft. per minute.

The joints between the tilting hearth and the stationary ends of the reheating furnace are packed with asbestos, which closes them tightly and will endure the moderate temperature, say 1,350° C., fairly well.

The metal is a little hotter than when it left the Pernot furnace, but is still of an orange heat. In the molds the metal is puffed up by gases being evolved, but settles more solidly soon after being cast, leaving a fin standing up all around each pig. For casting into this form, an extra charge is made.

Yield.—The yield of washed metal is about the same as the weight of the original crude iron charged.

Chemistry of the Washing Process

The crude iron contains: Silicon, 1 to 1.25 per cent.; sulphur, 0.02 to 0.03 per cent., and phosphorus, 0.090 to 0.10 per cent. All of the silicon and manganese, 90 to 95 per cent. of the phosphorus, and 30 per cent. of the sulphur are eliminated. The silicon and manganese with 75 to 80 per cent. of the phosphorus leave the iron in 10 min. and 30 per cent. of the sulphur in 15 min. The silicon, manganese, and phosphorus are removed by oxidation, while the sulphur seems to be eliminated by liquation, the sulphide of iron getting into the slag mechanically without oxidation. At certain high temperatures, the affinity of sulphur for oxygen seems to be weaker than its affinity for iron. The slag takes sulphide of iron from the metal until equilibrium is established between the proportions of sulphide in the two, when no more will pass. When a second slag is formed, some of the sulphide will enter that, making the elimination from the metal more complete. Should the slag have too much sulphide, as has happened when some blast-furnace slag got in with the crude iron, some of the sulphide will pass from the slag into the metal, and in that way establish the equilibrium.

Individual heats of washed metal have been made having as little as 0.002 and even 0.001 per cent. of phosphorus, and it has been furnished on specification of 0.006 phosphorus and under.

When iron heavily charged with carbon, like washed metal, lies molten in contact with molten oxide of iron, there will be, if the heat is high enough, a continuous oxidation of carbon to CO until the temperature falls to a certain degree, when the action will cease. That reaction is endothermic and the consumption of heat automatically brings the temperature down to that point. This temperature has not been determined, but there is ground for suspecting it to be near 1,300° C.

Analyses of Metals and Slags

Typical analyses of crude irons with washed metals and slags therefrom are given below.

	Pig Iron, Per Cent.	Washed Metal, Per Cent.
Silicon	1.25	none
Phosphorus	0.095	0.010
Sulphur	0.020	0.015
Manganese	.....	none
Combined carbon	.....	3.40

Final Slags

	Per Cent.
SiO_2	16.50
MnO	6.50
CaO	8.50
P_2O_5	1.05
S	0.075
FeO	50.25
Fe_2O_3	16.90
	99.775

The Removal of Metalloids in the Pig-Washing Process

BY EDWARD L. FORD, YOUNGSTOWN, OHIO.

In the accompanying diagram (Fig. 3) showing the removal per cent. of metalloids in washing, the rate of elimination of each seems to be constant for about the first 10 min., and (with the possible exception of sulphur) the angle which the line of each makes with the base line probably represents the relative net heat results of each complete reaction. I use the word "complete" because the net result is the sum of a number of things: In the case of silicon, for example, the heat + or − from Si to SiO_2; from FeO to Fe; from Fe_2O_3 to FeO; the formation of silicate by the silica; the liquefying of this silicate, etc.

Why these rates should not remain constant up to complete elimination is due to a number of things—changes in temperature, changes in character of atmosphere, changes in composition of slag and resultant metal, and very possibly other causes. We have worked out some of these; and, although they may not be the only factors, they seem to explain fairly well what happens.

Silicon.—The rate of silicon removal is constant to about 95 per cent., when it begins to become less rapid. This, I think, is due entirely to the want of perfect mobility of the slag and metal, and therefore to the fact that the silicon is not present in equal amount in all parts of the metal at this stage of the process. In fact, it must take an appreciable amount of time for the metal in the center of the mass to reach iron oxides on its surface.

Manganese.—Manganese seems to behave in the same way, although the rate of removal is a little less rapid. Its retardation at the end is due to the same causes.

Phosphorus.—The phosphorus removal-rate is constant to about 70 to 75 per cent., when retardation sets in, and progressively increases, the line tending to parallel rather than intersect the 100 per cent. line. The long time required to remove phosphorus, as compared with silicon and manganese, would make the lack of mobility at this stage a factor of practically no consequence; but, after the elimination of silicon, carbonic oxide begins to be formed; and (as Pourcel showed very clearly, in the early days of the basic Bessemer process, in the discussion on the necessity for the overblow to remove the phosphorus) phosphorus cannot be removed from iron when CO is being formed intermolecularly in quantity; and no doubt the proportion of phosphorus removed is inversely related to the amount of CO formed, during a given period. True, the forma-

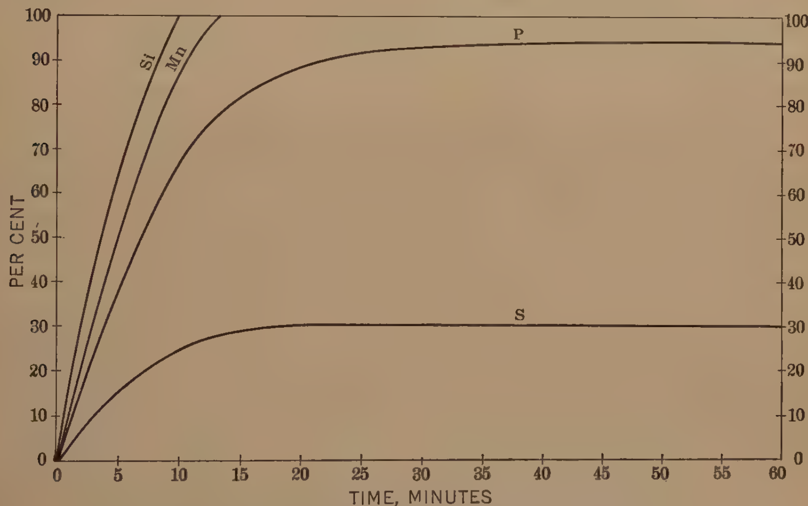

Fig. 3.—Removal of Metalloids in the Washing Process.

tion of CO, by lowering the temperature of the bath, tends to stop itself; but it completely ceases only when the metal is so cool that it begins to set; and this, of course, impairs the fluidity necessary to bring new surfaces of metal in contact with the slag, in order to oxidize the small amount of phosphorus left. We, therefore, finally reach a point of equilibrium when the CO formed just reduces the amount of phosphorus oxidized. If we increase the heat of the furnace, to increase the fluidity of the metal, we increase again the amount of CO that can form, and so are no better off.

Sulphur.—Our theory of the elimination of sulphur I have never seen formulated and would hesitate to report it here, if I were not trying to give the *rationale* of the washing process, as we have known it. Even at quite a low temperature, sulphur appears to form sulphides, preferably

to oxygen compounds; and as the temperature increases, this tendency seems more marked. In neither the blast furnace, the converter, the open hearth, or the puddling furnace, nor in our washing process, is the sulphur removed, except as a sulphide. In roasting sulphurous iron ores, however, it does go off as SO_2; but this process, as is well known, is performed at as low a temperature as possible. Our view is that for any given temperature, atmosphere, slag, and metal composition, the amount of sulphide in the slag bears a constant ratio to the amount in the metal. That there is an equilibrium, so to speak, which cannot be altered without a change in one, at least, of the controlling factors above mentioned. In our own process, as carried out by us, the ratio seems to be one in the metal to five in the slag, or, to put it another way, a slag with 0.10 per cent. sulphur will give a metal with 0.02, and to get 0.01 per cent. sulphur in the metal, the slag must not have more than 0.05 per cent. As corroborating this proposition, the two following incidents may be interesting:

On one occasion, pig iron with 0.025 per cent. sulphur was run into the washing furnace at a time when an accident at the blast furnace had let some blast-furnace cinder get into the receiver together with the pig metal. Some of this cinder ran with the metal into the washing furnace. The resulting washed metal carried 0.04 per cent. sulphur. With our ordinary practice this would have been about 0.017 per cent.

Some washed metal with 0.015 per cent. sulphur was melted in a cupola with coke. The resultant metal ran 0.10 per cent. sulphur. Some white iron with 0.10 per cent. sulphur was melted in the same cupola. Result: sulphur, 0.11 per cent. We had on our yard a heat of washed metal, containing 0.09 per cent. sulphur. This was melted in the same cupola. Result: sulphur, 0.12 per cent. These results seemed to point to from 0.10 to 0.12 per cent. as the equilibrium point in the resultant metal from these operations, no matter what sulphur the metal contained to start with.

With the above statement of our sulphur theory, the following is our explanation of the sulphur line in the diagram:

This sulphur line probably has only this relation to those of the other metalloids, that it tends to become tangent always directly below the point of complete silicon elimination, because at this time ends the formation of new sulphur-free slag, capable of taking up more sulphur from the metal. The percentage removal, therefore, varies with the sulphur contents of the iron, in contradistinction to the other metalloids, the percentage removal of which is constant.

Low-sulphur washed metal is the only marketable product; hence we aim to keep the sulphur in our pig iron as low as we can, with low-silicon iron. Our "fix" ores contain a little sulphur; the gas from our producer carries a little; and our limestone a little; so that the resulting slag, if none

came from the pig iron, would run about 0.03 to 0.04 per cent. as a minimum. Our practice shows us that, in a metal containing 0.007 per cent. sulphur, this would give us no removal.

If we could obtain raw slag materials absolutely free from sulphur, our washed metal would be 0.007 per cent. lower in sulphur with our regular pig iron than it is now. If we were satisfied with 0.05 per cent. sulphur in our washed metal, we could use a pig so high in sulphur that the amount taken out would probably approximate 60 per cent., rather than 30, as now; for when we formed our second slag, it would be capable of taking up a considerable portion of this remaining sulphur, before reaching the equilibrium point.

If the other conditions incident to the process permitted it, we could, by making new slags, reduce any amount of sulphur in the pig to about 0.01 per cent., but we could never get much lower than that, no matter how low the pig was, as long as our slag-making materials were as high in sulphur as they are now.

Our experience teaches us, however, that the blast furnace is the best and cheapest desulphurizer.

Modern Development in the Combustion of Blast-Furnace Gas with Special Reference to the Bradshaw Gas Burner

BY K. HUESSENER,* PITTSBURGH, PA.

(New York Meeting, February, 1916)

Introduction

This paper attempts a survey of the principles involved in the combustion of blast-furnace gas in boilers and stoves. I do not expect to be able to give much information which is actually new, since the laws and methods of calculation pertaining to this problem are contained in a number of handbooks; but, so far as I am aware, they have nowhere been treated exclusively with reference to this one problem of gas economy. The road which I follow is less that of strictly scientific research than that of the practical engineer, faced by operation problems which require a quick solution by easily accessible means without a prolonged search in scientific handbooks. For this purpose I give a number of tables and diagrams, as well as the results of many years' practical working experience as combustion engineer, which I trust will prove useful to my readers.

Historical

Considering the tremendous efforts made during the last 15 years to increase the efficiency of all kinds of power plants, it is highly surprising that until a comparatively very recent period, progress in the economical combustion of blast-furnace gas has been so slow.

The explanation generally put forth, that fuel has been so cheap that it was hardly worth while to bother about this economy, will not hold good, for the greatest trouble was taken to improve the combustion of coal and to reduce the consumption of steam. The most plausible explanation appears to be the general impression, held as an accepted fact, that gas, especially hot, uncleaned, blast-furnace gas, was not a good boiler fuel. Many managers of high repute had tried unsuccessfully to improve the conditions of its use, and the dictum went forth that efficiencies of over 60 per cent., particularly with high loads, were impossible. This was the situation not only in the United States, with very cheap fuel, but also in Europe, where fuel prices are twice and three times as high.

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The failure to improve conditions was, to a large extent, due to the type of boilers mostly in use up to the last five or six years. This also is true for both America and Europe. In America, one-pass boilers of the Cahall, Cook and Wheeler types were used in large numbers, and the boiler capacity was not always sufficiently large so that overload was frequently necessary. While with up-to-date combustion arrangements fair efficiencies can be obtained at the present time from these one-pass boilers, they have baffled at least all my efforts to secure that result with an overload.

In Europe 90 per cent. of all blast-furnace boilers were internal-flue horizontal boilers, and the gas was not cleaned, dirt accumulated in the flues and rendered good efficiencies impossible, however good the combustion itself might have been.

In the case of water-tube boilers it was of course possible to remove the dust by the use of soot blowers or steam- or air-blowing lances. This could not be done with the internal-flue boiler, where in order to remove the dust it was necessary to lay off the boilers altogether. With this type of boiler both efficiencies and loads declined continuously, and after one week's working it was impossible to run a boiler on rated load. The natural expedient was to clean the gas; and accordingly in the beginning of the century a number of efficient gas-cleaning plants were developed in Germany. The cleaning of the gas rendered the use of blast-furnace gas in gas engines possible. At a time when cleaning plants, which formed so large an item in the total expenditure over a gas-engine installation, were anyhow considered indispensable and furthermore when an efficiency of 50 per cent. on the gas-fired boilers was considered the best possible obtainable (as was the case 15 years ago), the superiority of the gas engine was hardly open to discussion. In this connection it must be kept in mind that during the first years of gas engines, it was generally accepted that they would not consume more than 14,000 B.t.u. per kilowatt-hour. At this time the best steam engines would still consume 20 lb. of steam per horsepower, equivalent to 26,000 B.t.u. per kilowatt-hour. If then a kilowatt-hour had to be raised at an efficiency of 50 per cent., it meant a consumption of 52,000 B.t.u. for the steam engine, as compared with 14,000 B.t.u. for the gas engine. Since most people considered the gas-fired boiler at that time a thing of the past, little effort was spent for a number of years in improving the combustion arrangement. This development was also held back because the gas-cleaning plant by itself brought about, with the internal-flue boiler, a marked improvement both in efficiency and load, so that an efficient combustion was no longer the all-important factor.

After the gas engines had been in use at a number of places, however, they were found not to be an unqualified blessing; they gave high efficiencies only with high and constant loads, which in iron and steel

works the blowing engines alone can furnish, but which could not be maintained in mill work. In 1907, Mr. Hoff, a German iron-works manager of high repute, investigated 32 power plants using gas engines at various German steel works and ascertained that the average load of all of them was only 52 per cent., that, in one instance, it was as low as 32 per cent.; and that it had not been possible in any case to generate a kilowatt-hour with the low consumption of 14,000 B.t.u., a claim put forth by the makers of gas engines.

Today it is generally accepted that, even at so-called "mixed" power plants, where only the constant load is carried by the gas engines, and the excess load by steam turbines, 18,000 B.t.u. are required for each kilowatt-hour. Moreover, since for gas engines the gas has to be cooled down to atmospheric temperature, we must add to the 18,000 B.t.u. actually consumed the sensible heat which has been taken out of the gas in the washing process; and as this sensible heat represents 8 per cent. of the latent heat, the actual heat consumption of the gas engine under the most favorable conditions stands today at 19,400 B.t.u. per kilowatt-hour. In this country, the General Electric Co. claims that with its steam turbine it can raise a kilowatt-hour with an actual heat expenditure of 19,000 B.t.u.

During this development of the gas engine, great strides were made in the improvement of the steam engine by the advent and development of the steam turbine. The 20 lb. of steam formerly required per horsepower, was gradually reduced to 14 lb. and, in some cases, where very large superheats were used (in Germany one is going as far as 450°F.) to 11 lb. per horsepower.

This concurrent development of steam engine and steam turbine brought about a revival of interest in the perfection of combustion. In the meantime, the waste-heat coke oven, which, up to about 1900, was almost universally used in Germany, gave way to the regenerative coke oven, but this development was generally retarded by the conviction everywhere held that gas was not suitable for boiler firing, and that, unless it could be used in gas engines or sold to gas works, the adoption of the regenerative coke oven would always mean a sacrifice in steam.

The comparative figures in this respect put forth as recently as 1912 by Baron Coppée at the British Iron and Steel Institute Meeting in Brussels were 0.75 ton of steam per ton of coal from the regenerative oven, as against 1 ton of steam per ton of coal from the waste-heat oven.

Burners

It is a remarkable fact that in Germany the first efficient gas burners were developed for coke-oven gas, although this kind of gas did not represent nearly as important an economical factor as the blast-furnace gas.

This is beyond doubt partly explained by the fact that coke-oven gas was always available in a thoroughly clean condition, so that dust troubles were not encountered. After the question of economically burning coke-oven gas had been solved, the inventors of these burners turned their attention to blast-furnace gas, cleaned and cooled blast-furnace gas being available at a large number of plants. As the difficulty of dirt accumulation in the internal-flue boiler cannot be overcome, even at the present time, it is still held in Europe that for the purpose of obtaining good boiler efficiencies it is necessary to clean the gas.

In both design and appearance, nearly all of the first German burners followed the Bunsen type, and were mostly round in shape, partly, perhaps, because that shape was obviously the proper one for the circular flues of the internal-flue boilers. The chief disadvantage of the cylindrical burner, namely, the varying friction with the varying gas pressure, did not figure very largely, since, owing to the cleaning plants (which as a rule almost entirely absorb the already low top pressure in European plants), "boosters" to restore the pressure of the gas were necessary anyhow, and provided a fairly uniform gas pressure. The second disadvantage of the Bunsen type of burner, namely, that the number of units varies with the size of the boiler, was of much smaller importance in Europe, since the small (usually 100 to 150 hp.) boilers did not as a rule call for more than two burners. Developments in the United States differed materially from those in Europe. In the first instance, the boilers almost universally used here are water-tube boilers of much larger size, and were all originally equipped with two burners, so that the natural tendency of the American blast-furnace operator and steam engineer was to adhere to the two burners.

The Bradshaw Burner.

A. N. Diehl[1] has described a number of types of American burners, so that I can content myself with a few words about the development of the Bradshaw burner, to which this paper, among other things, is devoted.

G. D. Bradshaw realized from the beginning the difficulty of pressure differences, which would result in differing amounts of friction, and likewise differing percentages of air aspiration. While nearly all the other burners have flat or tapering mixing tubes, he followed the Venturi principle and gave the burner the general shape of a Venturi tube. This principle is as follows:

In the flow of any fluid in a pipe or stream the sum of the static or pressure head and the dynamic or velocity head at any two sections is a

[1] A. N. Diehl: Modern Methods of Burning Blast-Furnace Gas under Stoves and Boilers, *Monthly Bulletin*, American Iron and Steel Institute, vol. iii, No. 10, p. 265, (October, 1915).

constant, if the friction and eddy-current losses between the sections be neglected. That is,

$$h_1 + \frac{v_1^2}{2g} = h_2 + \frac{v_2^2}{2g}$$

where h_1 and v_1 are the pressure and velocity at the first section and h_2 and v_2 the same at the second section. If h_1 be practically the atmospheric pressure, h_2 may be made less than the atmosphere by a proper and not prohibitive change in section between the two points. When a burner embodies this principle, the subatmospheric pressure h_2 may be employed to draw air into the stream of gas. Should the quantity of gas change, the difference between the gas pressure and the atmospheric

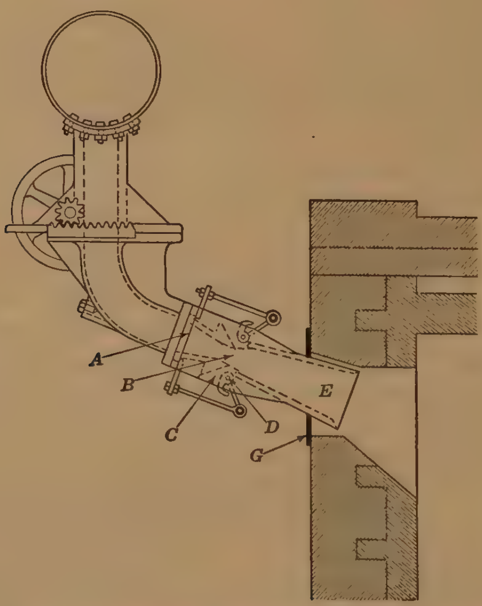

FIG. 1.—CUT THROUGH BRADSHAW BOILER BURNER.

pressure increases in proportion to the difference in the squares of the former and the new velocities. Since the flow of gas through an orifice is proportional to the square root of the pressure difference, the air drawn in by the suction varies directly with the gas flow, and, once adjusted, the ratio of gas and air is kept constant. All of this is predicated upon a reasonably constant total head, including both pressure and velocity heads. The total head at any point may be kept closely constant by preventing great changes in the pressure of the gas and the draft in the combustion chamber. The gas pressure will vary with the irregular operation of the furnace, but the short path of the gas through the burner and its small losses therein make the combustion draft by far the greater

factor in the maintenance of a constant condition within the burner itself. This draft can be held constant by any accepted type of balanced draft regulator.

Fig. 2.—Stirling Boiler with Bradshaw Burner.

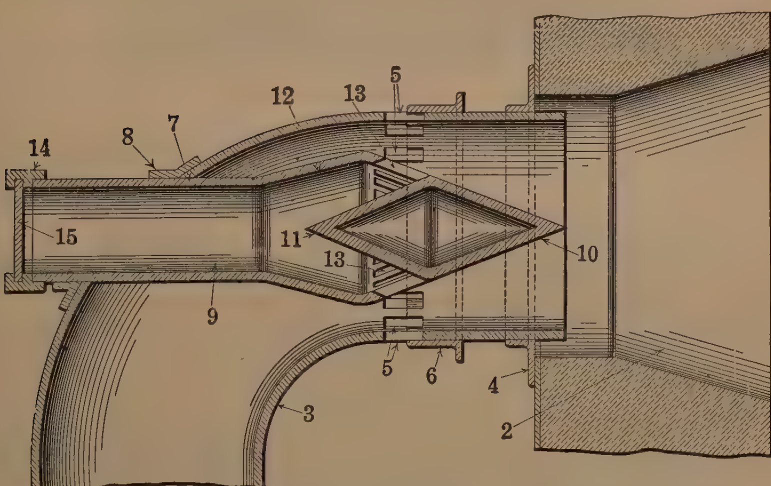

Fig. 3.—Cut through Bradshaw Stove Burner.

Figs. 1 and 2 clearly show the general principles of the Bradshaw boiler burner. The gas enters at *A* into the converging section of the burner by which its velocity is gradually increased as it approaches the throat *B*. At *B* are placed the air-inlet openings *C*, which are regulated

by the adjustable scroll dampers *D*. From this point onward the burner is diverging and the velocity of the mixture of gas and air is reduced by the expanding tube *E*, which ultimately discharges into the combustion chamber. At *G*, additional secondary air enters around the tube *E*. The burner is designed in such a manner that the velocity of gas and air in the expanding tube *E* is sufficiently high to prevent back flashing. Should a large reduction in the quantity of gas cause back flashing, the return of the gas supply automatically forces the flame back out of the discharge tube.

In the stove burner, where the round shape had of necessity to be adhered to, Bradshaw secured the operation of the same principle by inserting a bullet-shaped cone in the inside of the burner tube, which also gives him the converging entrance passage of the gas and diverging outlet passage therefrom to the outlet part, as shown in Fig. 3.

Fig. 4.—Bradshaw Stove Burner Bullet.

The Bradshaw stove burner is a reversal of the boiler burner in this respect: that the primary air is admitted internally through the air tube 9, as well as externally through the holes 5, shown in Fig. 3. The air regulation is obtained by slide 15 and the annular ring 6. Fig. 4 shows one of the parts of the Bradshaw stove burner. It will be noticed that in other respects both the shape and the general design of the present stove burner are adhered to.

Principles of Combustion

The principles of combustion are of course generally known; but I insert them here as an introduction to a few useful tables concerning the composition of waste gases. The combustion of gas, as everybody knows, is the chemical reaction of the combustibles in the gas with the oxygen contained in the combustion air. Each kind of gas requires a certain definite quantity of oxygen for complete combustion. If too little air is admitted combustion must remain incomplete; as a matter of fact, it

will nearly always be impossible to obtain complete combustion for any length of time, without a certain amount of excess air, since the mixture of gas and air is rarely if ever sufficiently intimate to effect complete combustion unless an excess of oxygen is present. In any case it is not safe to burn the gas with less than from 5 to 10 per cent. excess of air, because the variation in the composition of the gas renders the danger of incomplete combustion too great.

An excess of air is also necessary, because combustion without such an excess of air, particularly if all the air is primary air, has a tendency to be explosive, and creates eddies which interfere with the aspiration effect of the burner.

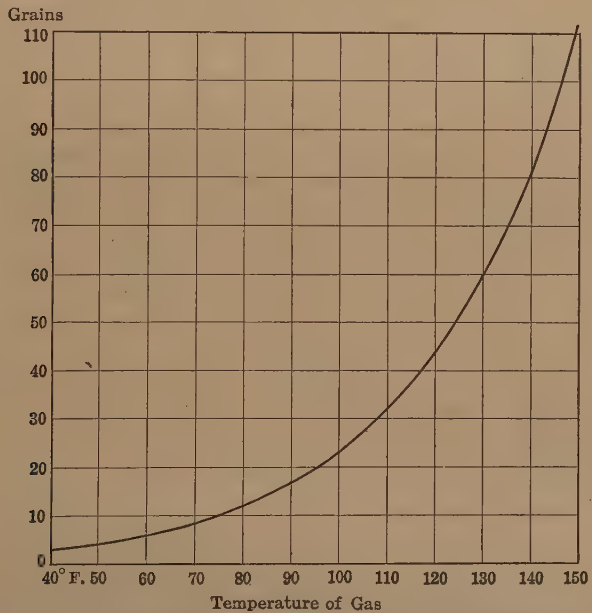

Fig. 5.—Grains of Vapor per Cubic Feet of Dry Gas Measured at 60°F. and Atmospheric Pressure in Saturated Gas at Different Temperatures.

The volume of oxygen necessary in order to burn completely a given volume of gas is found as follows:

$$\text{Total oxygen} = \frac{(CO + H_2)}{2} + 2CH_4$$

I mention these three constituents as they are the only ones ever present in blast-furnace gas. The composition of the waste gas providing complete combustion without excess air will then be:

$$CO_2 = CO_2 + CO + CH_4$$
$$H_2O = H_2O + H_2 + 2CH_4$$
$$N_2 = N_2 + \frac{\text{Total Oxygen} \times 79.5}{20.5}$$

The composition of the waste gas from blast-furnace gas containing 25 per cent. CO, 13 per cent. CO_2, 3 per cent. H_2, 0.4 per cent. CH_4, 58.6 per cent. N_2, when burnt without excess of air will then be as follows:

38.4	parts CO_2
3.8	parts H_2O
116.0	parts N_2
158.2	parts

The theoretical CO_2 on dry waste gas will therefore be 24.9 per cent. As with complete combustion without excess of air each cubic foot of gas will yield 154.4 cu. ft. of dry waste gas of which 24.9 per cent. is CO_2, it is simple to calculate the actual volume of the waste gas from the percentage of CO_2 present.

It will be noticed that in all these calculations the furnace gas has been assumed to be dry. Since, however, fully dry blast-furnace gas never exists, it is in each instance necessary to ascertain the amount of moisture in the gas. This is done by finding the dew point of the gas, that is, the temperature at which the water commences to condense. Fig. 5 shows the amount of water vapor per cubic foot of dry gas, measured at 60°F. and atmospheric pressure, in saturated gas at different temperatures.

Causes of Low Boiler Efficiency

There are four causes of low boiler efficiency which I will review in the order of their importance: (1) incomplete combustion; (2) combustion with large excess of air; (3) water vapor in the gas; and (4) deficiencies in the boiler plant itself.

The first three can be combined in one by saying "low combustion temperature." Deficiencies of the boiler plant itself (for instance air leakages in the front part of the boiler) are also frequently the cause of low combustion temperatures. Nevertheless, high combustion temperatures are quite compatible with low efficiencies due to faults of the boiler plants, such as cracks in the rear of the boiler settings, broken baffles, scale, dust accumulations on tubes, etc.

Low Combustion Temperature

It is well known that every gas has a certain highest combustion temperature which is obtained if the gas is fully dry and burned without excess of air. This temperature lies for most blast-furnace gas between 2,450 and 2,500°F. for cold gas, and between 2.8 and 11.8 per cent. higher for hot gas of a temperature of 200 to 600°F. It must be the aim of the gas engineer to get as near as possible to these theoretical combustion

temperatures. If the combustion temperatures are right, the rear temperatures will take care of themselves. It is an accepted rule that for

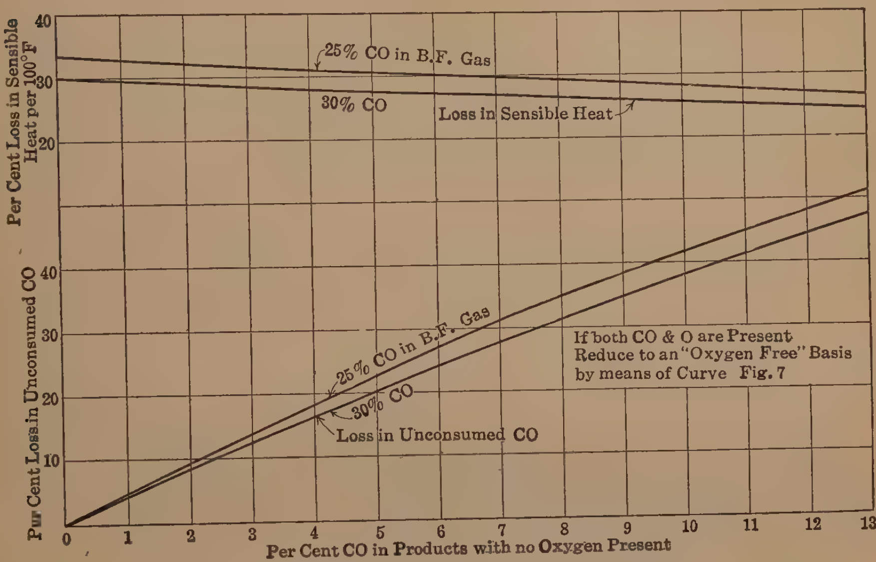

Fig. 6.—Losses through Incomplete Combustion.

any given gas (provided the same quantities of gas are passed in a given time unit) the rear temperature will always be in an inverse relation to the combustion temperature. This will show at once the importance of

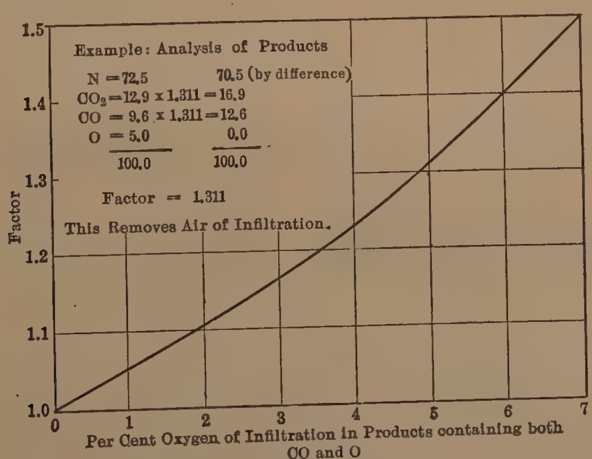

Fig. 7.—Factors for Correcting Analyses.

combustion with a low excess of air as a large excess will not only increase the volume but also the temperature of the waste gas. All this is predicated on otherwise faultless boiler conditions.

Fig. 6 shows in what manner the efficiencies are affected by unburned CO, giving the losses of both sensible and latent heat through incomplete combustion, for gases containing 25 and 30 per cent. CO with 0 to 13 per cent. CO in the waste gas, without excess of air.

If the analysis of the waste gas shows excess of oxygen in spite of the unburned CO, both CO_2 and CO must be multiplied by a certain factor which depends on the amount of oxygen in the waste gas. Fig. 7 gives these factors for amounts of oxygen varying from 0 to 7 per cent.

		Per Cent.
Example: the analysis of waste gas is	N_2	74.8
	CO_2	20.6
	CO	0.6
	O_2	4.0
		100.0

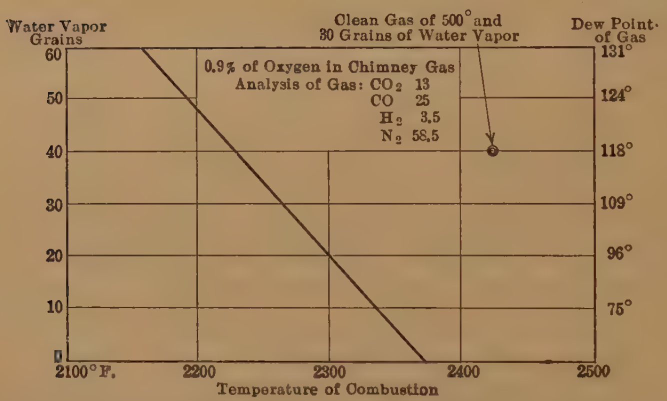

Fig. 8.—Temperatures of Combustion of Blast-Furnace Gas Burned with Air at 60°F. and 10 Per Cent. Air Surplus. Gas Saturated with Different Amounts of Water Vapor per Cubic Foot of Dry Gas Measured at 60°F. and Atmospheric Pressure.

For 4 per cent. O_2, the factor is 1.23 and by multiplying CO and CO_2 by this factor we obtain the following oxygen-free analysis:

N_2 =		73.924 (by difference)
CO_2	20.6 × 1.23 =	25.338
CO	0.6 × 1.23 =	0.738
O_2	=	0.0

The loss due to the unburned CO can then be ascertained from Fig. 6 as 2.8 per cent. for gas with 30 per cent. CO, and the total loss through sensible heat 2.98 per cent. for each 100°F. rise in temperature over boiler-house temperature. If stack temperature is 500°F. higher than

the boiler-house temperature the total loss is 17.7 per cent. This table is correct for all gas containing $CO + CO_2$, 38; H_2, 3; and N_2, 59 per cent., the dew-point being 120°F. and the gas temperature 400°F. If used for other gas it must only be considered a rough-and-ready way for ascertaining the approximate stack losses.

Fig. 8 shows how the combustion temperature is affected by water vapor in the gas. It gives the temperature of combustion of blast-furnace gas burned with air at 60°F. and 10 per cent. air surplus, and carrying different amounts of water vapor per cubic foot of dry gas measured at 60°F. and atmospheric pressure.

This curve is particularly interesting, since it answers the question, to what temperature cleaned blast-furnace gas should be cooled, in order to give the best efficiency. It will be admitted that without exceptionally cold water the gas cannot be cooled below 75°F., but even this temperature will not often be obtained, as in most instances it is preferable to use the cooling water from the blast furnace for washing purposes so that the possible temperatures will usually lie between 80 and 100°F. The difference in combustion temperature will not be greater than about 40°F. or less than 2 per cent. The heat abstracted by the water vapor in the waste gas is also trifling. Assuming the waste-gas temperature to be 600°F., then the loss as compared with the gas of 70°F. will be as follows, allowing for the gain through sensible heat of the gas at the high temperatures:

Gain in efficiency as compared with gas of 70°F.	
For 80°F. gas temperature	0.06 per cent.
For 90°F. gas temperature	0.08 per cent.
For 100°F. gas temperature	0.03 per cent.
Loss of efficiency as compared with gas of 70°F.	
For 110°F. gas temperature	0.10 per cent.
For 120°F. gas temperature	0.34 per cent.
For 125°F. gas temperature	0.56 per cent.

This shows that as long as the gas temperature is kept approximately at about 100°F. the best possible results will be obtained and that it does not pay to use enormous quantities of water in order to reduce the gas temperature below this point.

Sensible Heat.—A few remarks will here be opportune about the importance of the sensible heat of the gas. The opinion has frequently been propounded that cold, clean gas is always preferable to hot unwashed gas. This is true for stoves which cannot be cleaned while they are working and on which hot uncleaned gas cannot be used to best advantage. If here gas is burned with a small excess of air, the high combustion temperatures will bring about fusing of the dust and consequent slagging in the stoves. In the case of water-tube boilers, however, where a blowing of

the tubes about once in every turn will effectually remove the dust from the tubes the opinion is not correct.

In order to be in a position to ascertain the value of the sensible heat in the gas at a glance it will be useful to consult Fig. 9 which shows curves giving the thermal capacity of the various constituents of blast-furnace gas from 60 to 1,100°F. This curve together with some of those mentioned above has been kindly put at the author's disposal by A. Steinbart of the National Tube Co., Pittsburgh.

As an example, let us take a gas of the following analysis: CO_2, 13; CO, 26; H_2, 3; CH_4, 0.5; and N_2, 57.5 per cent. at a temperature

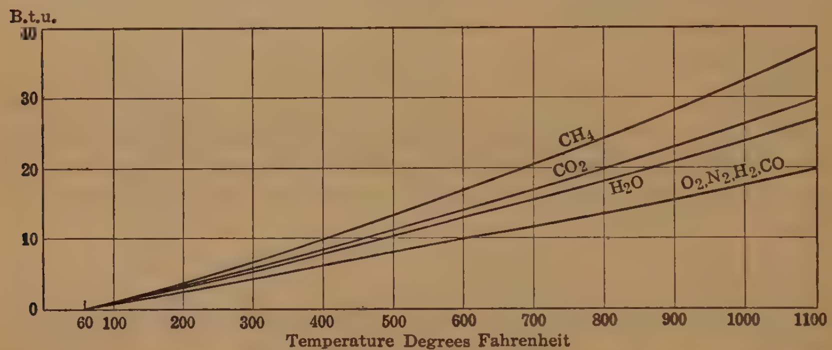

FIG. 9.—THERMAL CAPACITY OF GAS AT DIFFERENT TEMPERATURES.

of 450°F. By referring to the figure we find that each cubic foot of CO_2 contains 9.8 B.t.u., CO,H_2 and N_2, 7.2 B.t.u. and CH_4, 11.6 B.t.u. The total sensible heat is therefore as follows:

CO_2	0.13 × 9.8	1.274
CH_4	0.005 × 11.6	0.058
Balance	0.865 × 7.2	6.228
Total		7.560 B.t.u.

It cannot be sufficiently emphasized that a hot gas containing, say, 107 B.t.u., both in latent and sensible heat, is just as valuable as cold gas of 60°F. containing 107 B.t.u. in latent heat alone—at least as far as gas-fired water-tube boilers are concerned. The loss of sensible heat on account of washing is therefore a net loss which can in no way be recovered, even if we assume that the cost of washing the gas would be counterbalanced by the cost of cleaning the boilers. In a later part of the paper it will be shown that there is no difficulty in obtaining a boiler efficiency of 75 per cent. and more with uncleaned gas. A boiler efficiency of 75 per cent. on hot gas would correspond with an efficiency of 80.5 per cent. on cold gas and unless this could be exceeded, which I very much doubt, there could be no profit in gas washing.

Fig. 10 shows the losses through sensible heat on account of washing.

Fig. 11 shows how the combustion temperature is affected by the sensible heat when burned with an excess of air of both 10 and 50 per cent.

Results published by Mr. Diehl as regards the National Tube Burners at McKeesport are very interesting, as they show that high efficiencies are in exceptional cases compatible with low initial temperatures. I refer to the low-combustion temperature of about 210°F. They are in fact lower than they ought to be when the composition of the burned gas is considered. This composition, the same as the stack temperatures, leaves nothing to be desired. I explain these results by the fact that lingering combustion takes place and that the course of the products of combustion through the boilers is so unusually long that this lingering

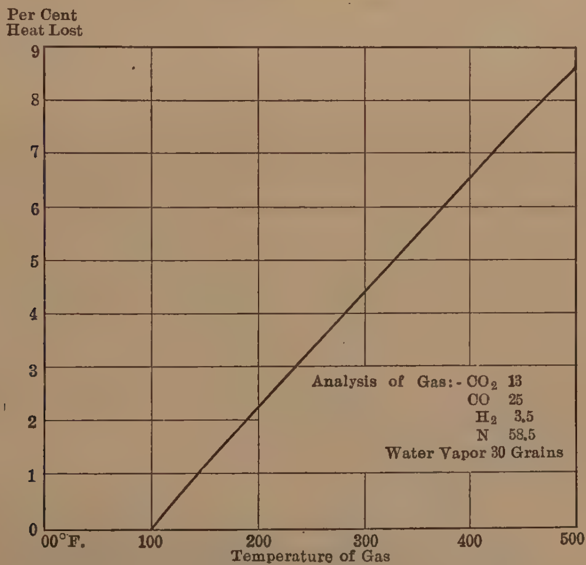

FIG. 10.—LOSSES THROUGH WASHING.

combustion which in 99 out of 100 cases is absolutely fatal to the efficiency of the boilers, does not make itself felt through a concurrence of exceptional circumstances which will rarely be repeated. In any case we have here the extraordinary fact of a burner which itself gives a comparatively low efficiency and yet results in a good boiler efficiency.

Water in Charge.—Hitherto we have only contemplated the effect of water added to the gas after it leaves the blast furnace. From the point of view of combustion alone, it is, of course, immaterial where the water is added, whether in the furnace or in the washing plant. Nevertheless, it will be interesting to note how a wet charge affects the gas economy of the furnace. Let us assume that the dew-point of a certain blast-furnace gas is 130°F. By referring to Fig. 5 we find that each

cubic foot of gas carries 60 grains of moisture. This water must not only be evaporated, but also brought to the top temperature of the furnace. Assume the latter to be 400°F. and that each ton of coke is responsible for 150,000 cu. ft. of gas measured at 60°F. and atmospheric pressure.

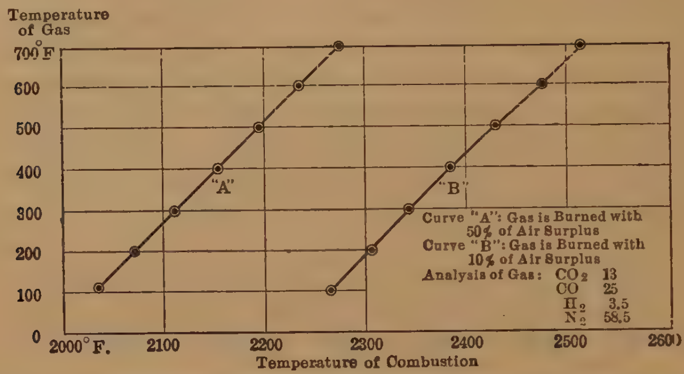

FIG. 11.—COMBUSTION TEMPERATURE WITH VARYING EXCESS OF AIR.

For each ton of coke 9,000,000 grains (1,285 lb.) of water must be evaporated and brought to a temperature of 400°F. These must first be raised to 212°F. which will consume 1,285 (212 − 60) = 195,100 B.t.u. The evaporation heat is 1,285 × 966 = 124,000 B.t.u. The 1,285 lb. of water

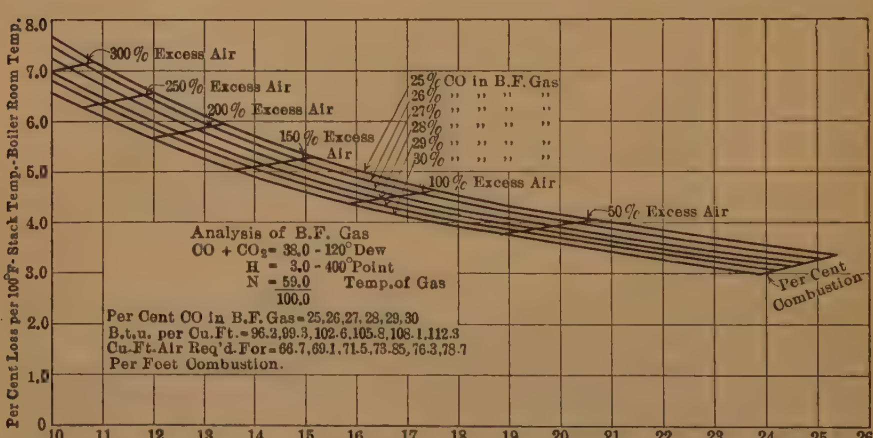

FIG. 12.—PERCENTAGE LOSSES THROUGH STACK GASES BASED ON CO_2 CONTENTS.

vapor are equal to 27,000 cu. ft. at 60°F. From Fig. 9 we find that the thermal capacity of water vapor from 212 to 400°F. is approximately 4.75 B.t.u. per cubic foot. This heat will be required to raise 1 cu. ft. to 400°F. so that 27,000 cu. ft. will require 128,500 B.t.u. in all. The total heat is thus:

	B.t.u.
To bring water to 212°F.	195,100
To evaporate water.	1,240,000
To heat vapor to 400°F.	128,500
Total, per ton of coke.	1,563,600

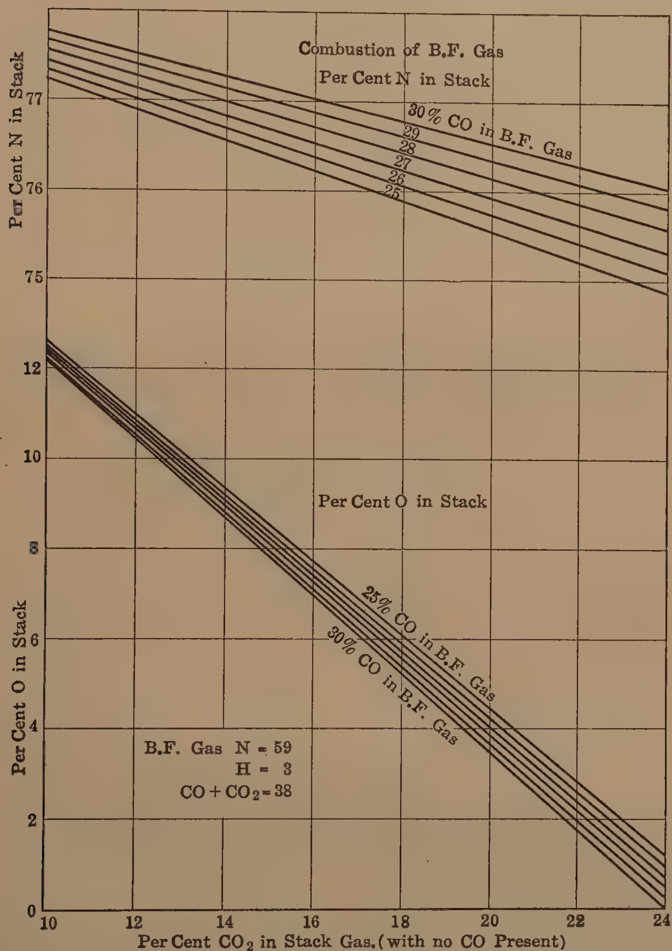

Fig. 13.—Oxygen and Nitrogen in Stack Gas Corresponding to Various Percentages of CO_2 for Complete Combustion.

If the calorific value of coke is 12,500 B.t.u., then 125 lb. of coke out of every ton used is employed in heating and evaporating the water.

While it is impossible to eliminate all this moisture, and while its total elimination would result in disturbances in the working of the furnace, owing to high top temperatures, one ought to allow the top temperature to reach the highest point which is compatible with the smooth

working of the furnace and not unnecessarily reduce it by damping the ores. For the sensible heat in the gas is just as important a factor for steam raising as the latent heat. With the various types of sintering plants there is no difficulty in recovering the valuable constituents from flue dust. The increased quantity of flue dust due to a dry charge need not deter the operator. That a fairly dry charge can be run has been proved by J. S. Fraser, at Monessen, Pa., who, although using large quantities of Messabi ores, does not damp his ores when charging the furnace.

Other Losses.—To complete the diagrams which are necessary in

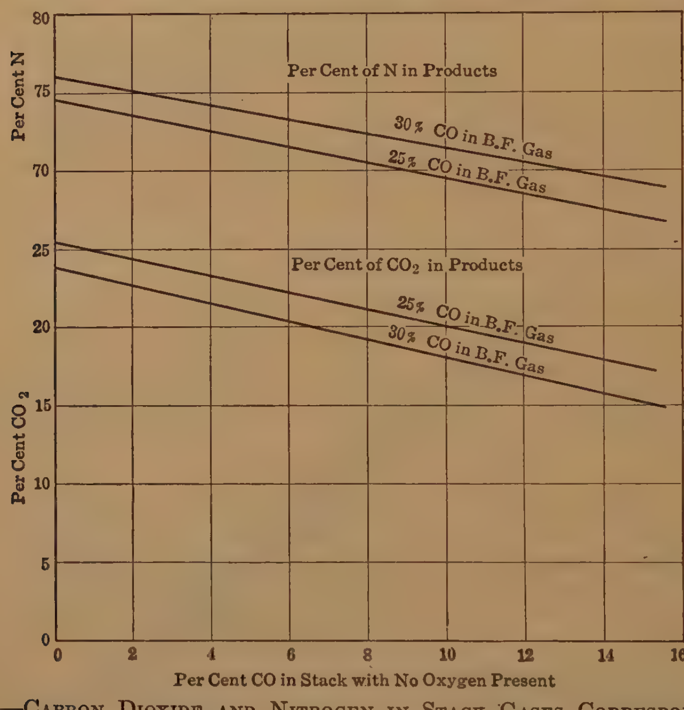

FIG. 14.—CARBON DIOXIDE AND NITROGEN IN STACK GASES CORRESPONDING TO VARIOUS PERCENTAGES OF CO_2 IN OXYGEN-FREE STACK GAS.

order to read at a glance the losses through waste gas as revealed by the stack analysis, I offer Figs. 12, 13 and 14.

Fig. 12 gives the percentage losses for each 100°F. rise of temperature over the boiler-house temperature, for varying CO_2 in the waste gases from furnace gases containing from 25 to 30 per cent. CO.

Figs. 13 and 14 are useful as checks on the correctness of Orsat tests. Fig. 13 shows the percentages of oxygen and nitrogen corresponding to the various percentages of CO_2 for complete combustion; Fig. 14 the nitrogen and carbon dioxide corresponding to the various percentages of CO for oxygen-free stack gas.

Bad Boiler Conditions.—Bad boiler conditions and the losses resulting

are: defective brickwork, causing air leakages; unsubstantial brickwork, causing high radiation and conduction losses; lack of boilerhouses, with the same results; defective baffles; scale; or too short passages for products of combustion. The remedies are obvious.

The whole secret of economy is to be able to know at any given moment what one is getting and what one ought to get. To start with the latter, blast-furnace operators cannot be too urgently advised to draw up once a week, or at least every month, a carbon balance sheet of their furnaces calculating the quantity of gas from the carbon in the furnace charge. If these balance sheets are to be of any value it will be necessary to take at least one gas analysis a day and also ascertain the dew-point of the gas every day; if the gas is washed, to keep reliable records of the gas temperature as well.

Air Infiltration.—How seriously boiler efficiency is impaired by air infiltration is shown by the following experiment made by Mr. Bradshaw at Johnstown. Mr. Bradshaw admitted false air into the boiler at various distances from the burner and before starting his experiment saw to it that he had the boiler and burners in otherwise perfect working condition.

Boiler No. 132, *March,* 1915

CO_2	Rear Temperature	
11.7	626°F	Gas pressure 4.5 in.
13.0	620°F	
13.3	616°F	
18.8	578°F	
18.8	575°F	
19.7	577°F	
15.0	580°F	Gas pressure 3.5 in.
18.0	545°F	

This experiment shows that air infiltration not only causes a loss, because all this air has to be heated to the stack temperature, but also results in an increase of the latter, or, in other words, reduces the heat absorption capacity of the boiler tubes. This is due to the fact that the thermal conductance is in direct relation to the difference of temperature between the gases and the outside temperature of the boiler tubes, so that if the temperature of the products of combustion is reduced, the conductance itself is reduced and less heat is transferred. This again shows how the boiler efficiency is affected by the combustion temperature; and this is also the explanation of the fact that in a good boiler 70 per cent. of the total evaporaton should be done in the first third of the boiler, 20 per cent. in the second and 10 per cent. in the last third. For this reason, air infiltration is a serious factor wherever it takes place and whether or not it interferes with the initial temperature. It is erroneous

to assume, for instance, that air infiltration which takes place in the rear passage will only tend to reduce the stack temperatures correspondingly to the amount of heat absorbed by the cold air. In a case where the stack temperature was 550°F. and the CO_2 18 per cent. with complete combustion (the low CO_2 being due to air infiltration in the rear of the boiler), the CO_2 after eliminating this air infiltration, was raised to 22 per cent. without an increase in the stack temperature. This proves that even at this point air infiltration means a net loss to the boiler.

Blowing the Tubes.—Another fact which must be constantly kept in mind is the necessity of blowing the tubes. Fig. 15 shows the stack temperature of a gas-fired boiler. The tubes were blown only once in

FIG. 15.—STACK TEMPERATURE CHART SHOWING EFFECT OF BLOWING TUBES.

24 hr. and it will be noticed that as a result of the blowing of the tubes the stack temperatures dropped by 150°. On account of these experiments, the tubes were blown every 12 hr. which resulted in a constant drop of temperature of 100°, a reduction by one-fifth of the loss through waste gas. An experiment made by the Pittsburgh Steel Co. illustrates how the load of the boiler is affected by the accumulation of dirt on the water tubes. When the first test was run the boiler tubes had not been blown for three days and the result was that the maximum average load obtainable from a 500-hp. boiler during 8 hr. was 720 hp. Subsequently the boiler tubes were carefully cleaned and another test of 8 hr. run the result being that instead of 720 hp., 827 hp. was obtained,

with even a decrease of losses through stack gas. In both cases the boilers were run at the maximum load compatible with good efficiency.

Boiler-Loads.—A few words as regards the boiler loads may be of interest. We, of course, all know that the official rating of 34.5 lb. per horsepower does not at all represent the actual capacity of the boiler. In this connection it may be interesting to say something about European customs. In England, water-tube boilers of the Stirling or Babcock-Wilcox type are rated for normal working conditions at 4½ lb. of actual steam per square foot of heating surface, which is equivalent to about 150 per cent. rating under the American rules, and the users of such boilers naturally expect to get a considerable overload out of them. In Germany, Belgium, and France, I do not think it would be possible to get an order for a water-tube boiler with a smaller guaranty than 6 lb. actual per square foot of heating surface, so that their rating really represents 200 per cent. of the American rating.

The results obtainable from internal-flue boilers are even more astonishing. I have run boilers of that kind at 8½ lb. actual per square foot of heating surface, equivalent to 9.75 lb. from, and at, 212°F., which would represent an overload of 282 per cent. for American practice. As a matter of fact, the flexibility of a modern boiler is very much greater than is generally assumed, and high efficiencies are quite compatible with loads of 200 per cent. and more. This load depends entirely on the chimney draft; and the limit is not reached until all the dampers are wide open and balanced draft is established in the combustion chamber, at which draft the boilers get all the gas and air which the stack can handle. A very simple and effective means to increase the boiler capacity, which has been resorted to in a good many cases, is to install induced draft. In Europe a number of modern boiler plants have been built without stacks, with fan draft only, and it has been possible to obtain astonishingly high boiler loads.

The maximum boiler load is, of course, to a certain extent dependent on the dust in the gas, and the limit is reached as soon as it is no longer possible to keep the water tubes clean by blowing once every turn. This limit will probably lie at about 180 per cent. At least the Cambria Steel Co. has been able, by blowing once every turn, to run its boiler plant for many months at a load of 165 per cent. while maintaining excellent efficiencies. This load can be exceeded without difficulty if the gas is cleaned. As a rule, the cleaning of boiler gas will only be resorted to if, owing to lack of space, new boilers cannot be installed, since it would be impossible to show any saving even if the boiler capacity should be increased by cleaning, say 20 per cent. Take the case of a 500-ton furnace. With suitable burners, a furnace of this size is capable of generating 5,100 rated boiler-horsepower per hour. To generate this power at 165 per cent. load, 3,100 hp. boiler capacity is required. If the load were raised by gas cleaning to 200 per cent., 2,600 rated boiler-horsepower would do the work, so

that a 500-hp. boiler would be saved. But the gas cleaning would result immediately in a drop in the evaporation of about 7 per cent. or 357 boiler-horsepower. At $25 per boiler-horsepower per year, the actual loss through cleaning would be approximately $8,900 a year which alone would be sufficient to cover the expenditure for an additional 500-hp. boiler quite apart from the cost of a cleaning plant which would require at least another $20,000.

Method of Measuring Economy.—There has been considerable discussion about the means of ascertaining whether or not one is getting good economical results. By some of the operators it is maintained that the only reliable method is to ascertain the boiler efficiency by stack analysis and temperature, and assume a certain percentage for radiation and conduction losses; other operators, and especially a number of steam engineers, hold that this method is all right for occasional tests, but does not answer in cases where large numbers of boilers are operated, and it is used to keep track of the actual results obtained from month to month and from year to year. These men, whose opinion I favor, hold that a monthly figure giving pounds of steam per pound of coke is the one and only reliable method of tracing results. The chief objection to this method of ascertaining the efficiency of a boiler plant is, that a large percentage of the gases are used for the stoves, and in a good many cases some of the gases are also used in gas engines. I hold that occasional efficiency tests are very misleading indeed. It is not very difficult to obtain good efficiencies for a given period of a few hours or days, covered by a test. The main difficulty is to maintain such efficiencies over an indefinite period. I make bold to say that with any kind of a burner, as long as it is possible to admit sufficient air for combustion, we can obtain excellent efficiencies for a period during which a boiler is under constant supervision and is continuously regulated, but we shall first find, that as soon as the supervision is relaxed (as it must be, once the test is finished) the efficiency will fall back forthwith; and second, that the ordinary type of burner will have a comparatively low maximum load, and the stack losses will be much larger with the ordinary type burner at a high load than with a modern type burner, such as the Bradshaw.

The contention that pounds of steam per pounds of coke does not give any reliable information on account of the gas used by the stoves can easily be met. In the first place, as the stoves get the first supply of gas and the boiler only the surplus, there is a very simple method of seeing to it that the stoves always get the same quantity of gas. All that has to be done is to put a pressure gage in the boiler house and instruct the boiler foreman to keep a uniform gas pressure at the stoves, which will bring about a uniform gas consumption. Better still take up stoves and boilers together and equip them both with suitable combustion arrangements, giving the highest possible efficiency. If that has been

done, the pounds of steam per pound of coke during any one month give the joint results of both stoves and boiler; and if they fall below that amount which corresponds to the coke ratage, then it must be found out by efficiency tests whether boiler or stoves are responsible for the reduction in the steam output.

If any part of the gas should go to gas engines, a measuring device should be installed and, in ascertaining the results which ought to be obtained, due allowance for the gas consumption of the gas engine should be made. As a rule it will even suffice to take the total output of electricity for the month and allow 18,000 B.t.u. per kilowatt-hour—or whatever the consumption may be in the case.

I reproduce here a table published by S. M. Marshall, late Chief Engineer of the Cambria Steel Co., showing the evaporations to be obtained from blast-furnace gas at the various rates of pounds of coke per ton of iron:

Coke per Gross Ton of Iron, Pounds	Cold Gas, Steam per Pound of Coke, Pounds	Hot Gas, Steam per Pound of Coke, Pounds
1,736	3.56	3.85
1,800	3.61	3.90
2,000	3.73	4.03
2,200	3.86	4.17
2,240	3.89	4.20
2,400	3.99	4.31
2,600	4.12	4.45
2,800	4.24	4.58
3,000	4.37	4.72

The steam is given in pounds from and at 212°F. Mr. Marshall figures on a very low stove efficiency of only 57 per cent. If this were raised to 75 per cent. the above figures would be increased by 8.95 per cent.

These figures conclusively answer in the negative the question, whether it pays to wash blast-furnace gas for boilers. Take the case of a 500-ton furnace using 2,000 lb. of coke per ton of iron. The loss through cleaning is 1/5 ton of steam per ton of coke. If the average value of a ton of steam is 25c., cleaning for boilers results in a loss of 5c. for every ton of coke used, quite apart from the cost of cleaning, which is only to a very small extent offset by the saving in wages for cleaning boilers.

Efficiency Tests.—If the above evaporations are not reached, efficiency tests on both stoves and boilers should be made. The losses sustained are made up in both cases of stack losses, and radiation and conduction losses. The stack losses may be in sensible heat alone if complete

combustion takes place, or in both latent and sensible heat if the combustion is imperfect.

Stack Losses.—The stack losses must be ascertained through stack-gas analysis and temperature measurements. The stack gases should show, under total absence of unburnt CO, from 22 to 23 per cent. CO_2 and the temperature, even for high boiler loads, should not exceed 550°F. These results, of course, are obtainable on good modern boilers only, although even with imperfect boilers, like Cook, Cahall and Wheeler, efficiencies of 70 per cent. and more, when running on gas, are obtainable in everyday practice.

With the assistance of Figs. 6, 7, 12, 13, and 14, the stack losses of both stoves and boilers can be easily ascertained. It should not be overlooked, however, that, particularly with bad combustion arrangements, the CO_2 is a much varying quantity, so that conclusions should be drawn only from a large number of analyses. It will, therefore, always be advisable to use a CO_2 recorder and take Orsat tests for unburned CO at least every 5 min. for several hours.

Where mixed coal and gas firing is used, a corresponding allowance for the water evaporated by the coal should be made. If there is any doubt as to the amount of water evaporated by the coal, special tests for the purpose of ascertaining the evaporation factor to be used can easily be arranged.

In this connection it may be pointed out that the practice of mixed firing is anything but desirable. Where the boilers are equipped with combustion chambers of sufficient size, and a good combustion arrangement is available, pilot firing, even in the case of the one-furnace plant, is not necessary. The Cambria Steel Co., for instance, has run 20 B & W boilers, equipped with Bradshaw burners and good-sized Dutch ovens, without pilot firing of any kind, in spite of the fact that these boilers received their gas from one furnace only. The heat stored in the combustion chamber was high enough to re-ignite the gas, even if it had stayed away for 10 min. or more, during a cast or change of tuyère. In this case, however, the furnace boilers were connected up to the mill boilers so that there was no danger of losing the steam pressure. At many of the one-furnace plants, this connection does not exist, and pilot firing cannot be dispensed with, since at any moment it may happen that the gas gives way and the power has to be supplied by coal exclusively. In that case, it should be kept in mind that the boilers are primarily gas-fired boilers and that all that is necessary is to maintain a small fire on the grate. It is essential, that after coal has been thrown on, the fire and ash-pit doors should be sealed up again, to avoid infiltration of air, and that where the boilers are equipped with blowers, these should only be used while actual coal firing is necessary, and sealed up at all other times. If for any reason the gas is not sufficient to carry

the load, it is advisable to spread the gas over a certain number of boilers which are not coal-fired except in emergency cases, and do the coal firing on other, separate boilers. In view of the very different conditions under which coal and gas have to burn, the advisability of this procedure is self-evident.

Boiler Construction.—We may inquire next, how boiler efficiency is influenced by the boiler construction as apart from the condition of the setting. It has generally been held that the one-pass boiler, such as the Cahall, Cook or Wheeler, is absolutely hopeless from the point of view of efficiency, and when I first approached the question of improving the efficiency of this type of boiler I did so with no little misgiving. I do not claim that I am in a position to say definitely to what extent a one-pass boiler can be worked economically; but, in experiments made at the Upson Nut Co.'s plant in Cleveland, I was surprised to find that boilers of this type could be run on a 100 per cent. load with very high efficiency. In some cases this was as high as 75 per cent., but all attempts to overload this kind of boiler with any show of efficiency proved futile, in spite of ample draft.

The chief disadvantages of the one-pass boilers are the great difficulty of keeping them air-tight and their very large radiation and conduction losses in comparison with their capacity. In any case, it is interesting to know that it is possible to run a boiler of this kind on gas at rated load with a good efficiency. Where, therefore, ample boiler capacity in one-pass boilers is available, it would certainly not pay to install modern boilers for the purpose of increasing efficiencies.

As regards boilers of the modern type, there can be very little doubt that, from the point of view of efficiency, it does not matter so much which type is used. Equally good efficiencies have been and are obtained on all types of boilers, provided the combustion arrangements are up-to-date.

Boiler House.—A very important feature in the operation of the gas-fired boiler, as we have seen already, is the "moral" of the boiler house. It would be a grave mistake to assume that the adoption of a good combustion arrangement alone would solve the difficulty. That will go a long way toward better efficiencies; but wherever efficiencies in excess of 65 per cent. are desired, the supervision of brickwork settings and the regulation of the draft are of paramount importance. We have already considered the great influence of air infiltration.

Draft.—The second factor which is of great importance is the draft. That a burner of the type which employs a preliminary mixture of gas and air provides its own combustion air, independently of the chimney draft, is only partially true.

The aspiration velocity of any burner is chiefly dependent on the chimney draft. Mr. Buessel, of the American Steel & Wire Co. of

Cleveland, has made some interesting experiments on an installation of Bradshaw burners at the Upson Nut Co.'s plant. He inserted a pipe, flattened at the end, into the air port of the Bradshaw burner and measured the aspiration velocity in this pipe by means of an anemometer. It is evident that owing to the friction in this pipe the figures obtained do not give positive information but they are invaluable as giving the relative velocities for various boiler drafts. With the same amount of gas in both cases, Mr. Buessel found that for a chimney draft of 0.35 in. the aspiration velocity in the air ports was 888 ft. per minute and that after reducing the chimney draft to 0.17 in. it fell to 713 ft. per minute. It is evident that a burner which receives sufficient air for a given quantity of gas at one aspiration speed must have suffered from incomplete combustion for the same quantity of gas at lower speeds. The aspiration velocity stands in a direct relation to the velocity in the mixing chamber, provided that all friction which tends to absorb the aspiration effect is eliminated, as in the case of the Bradshaw burner. In order to keep the relation between gas and aspirated air constant, the pull which is effected on the velocity of the gas and air mixture by the chimney draft must be kept constant, or the quantitative relation between gas and air will be upset. The importance of draft regulation will of course depend on the variation in gas pressure, so that, for instance, at a place where these variations are small, draft regulations will not play nearly as important a part as in places where there are frequent and large differences in gas pressure. Draft regulation is particularly important where it is desired to burn the gas with a very low excess of air; for, with dampers in a stationary condition, an increase in the gas pressure must of necessity result in incomplete combustion. I have found that it is not safe to raise the CO_2 higher than 21 per cent. without automatic draft regulation whereas 24 per cent. can safely be maintained with such regulation. From gas containing 25 per cent. CO the loss through waste gas with 24 per cent. CO_2 at a stack temperature of 560°F. is 17.75 per cent. while with 21 per cent. CO_2 it is 20 per cent. (see Fig. 12). This difference of 2.25 per cent. may not appear important, but it must be kept in mind that with a lower initial temperature the stack temperature in the case of 21 per cent. CO_2 will also, as likely as not, be higher.

Supervision.—Then there is the item of supervision. If the dampers are automatically governed, as for instance by the McLean governing device, and Bradshaw burners are used, then the sole duty of the boiler tender will be to watch the settings. He will not have to watch either burners or dampers. Without automatic damper governing, he will have either to do hand regulation or regulate the air valves for the purpose of maintaining an adequate quantity of air. Such regulation

of the air-valves, particularly in large plants would be practically out of the question.

As regards air-tight boiler settings, it may be useful to know that a very satisfactory coating is obtained by means of whitewash and cement. After several coatings have been given to the boiler, the settings will get a smooth surface. The falling off of the coating at any part will at once be detected and can be repaired as it occurs.

In addition to watching the draft, even with the most perfect combustion arrangement, it is necessary to check the composition of the waste gas and ascertain the temperatures from time to time. One or two analyses per week for the boilers will be quite sufficient and, at least with the Bradshaw burner, resetting of the air ports will as a rule only be necessary where a change in the iron is made, with a consequent considerable change in the composition of the gas.

Results with the Bradshaw Burner

In conclusion, I wish to say a few words about the results obtained with the Bradshaw burner. I give below a list of 52 stack analyses taken at the Cambria steel works, which show an average of 21.4 per cent. CO_2 under total absence of CO, a stack temperature of 581°F. and a boiler load of 165 per cent. As the gas in question contains 25 per cent. CO and the rise in temperature over the boiler house temperature is about 500°, the total losses through waste gas are 20 per cent., so that the boiler efficiency, even if the losses through radiation and conduction were as much as 5 per cent., would be 75 per cent.

I might mention that with boilers in a boiler house, it is extremely unlikely that the radiation and conduction losses would be as high as 5 per cent. As far as I am aware, no reliable figures of these losses are available at the Cambria Steel Co. I have personally conducted tests on 500-hp. boilers in England and found that the radiation and conduction losses of these boilers, which were in a boiler house, were less than 2 per cent. The German periodical, *Feuerungstechnik*, published in 1914 a table of experiments made on 12 Stirling boilers, working in various parts of the United States, which showed highest losses of 2.5 and lowest losses of 0.75 per cent. through radiation and conduction.

It may be of interest to know that the dampers were hand-regulated at the Cambria steel plant, which accounts for the comparatively low CO_2. What can be done with automatic damper regulation is exhibited by the two charts, Figs. 16 and 17, one of which shows the gas pressure, the other the CO_2 in the waste gas, as given by a CO_2 recorder. The correctness of this recorder was constantly checked, and unburned CO was not found in any of the samples.

With regard to Figs. 16 and 17 attention should be given to the uni-

Cambria Steel Co. Analyses and Stack Temperatures of Waste Gases

Boiler No.	CO_2	O	CO	N	Stack Temperature	
2–18	20.0	4.5	0.0	75.5	635	Tubes not blown for 24 hr.
2–18	21.0	3.4	0.0	75.6	650	
2–19	21.7	4.0	0.0	74.2	615	
11–126	23.2	2.0	0.0	74.8	445	February, 1915, load 145 per cent. of rating.
11–127	22.5	2.3	0.0	75.2	440	
11–138	21.2	3.2	0.0	75.6	525	
11–137	22.8	1.6	0.0	75.6	525	
11–136	21.6	2.8	0.0	75.6	515	
11–135	20.4	4.0	0.0	75.6	555	
11–129	21.0	3.5	0.0	75.5	515	
11–130	22.0	3.0	0.0	75.0	485	
11–120	21.5	2.3	0.0	76.2	580	
11–121	20.0	3.8	0.0	76.2	570	
11–138	21.4	2.6	0.0	76.0	515	
120	22.6	1.4	0.0	70.0	576	May, 1915, load 165 per cent. of rating tubes blown 12 hr. before.
121	20.7	1.2	1.1	76.8	619	
122	23.3	0.4	0.0	76.3	568	
123	22.6	0.4	0.0	77.0	499	
124	21.0	3.0	0.0	75.7	569	
125	20.8	4.2	0.0	75.0	537	
126	19.5	4.9	0.0	75.6	627	
127	21.6	1.8	0.0	76.6	580	
129	19.4	3.8	0.0	76.6	568	
130	21.4	0.6	0.2	77.8	610	
131	22.8	2.2	0.0	74.9	518	
132	22.2	1.8	0.0	76.0	565	
133	21.0	2.0	0.0	77.0	655	
134	20.2	1.0	0.0	78.7	610	
135	21.6	2.6	0.0	75.8	585	Continued May, 1915, load 165 per cent. rating tubes blown 12 hr. before.
136	22.6	0.8	0.0	76.6	542	
137	22.8	0.6	0.0	76.6	655	
138	19.8	3.6	0.0	76.6	585	
139	21.0	4.6	0.0	74.4	595	
Average of last 19	21.4	2.08	0.0	76.3	581	
120	23.2	0.8	0.0	76.0	557	June, 1915, tubes blown from 6–12 hr. before taking temperatures 165 per cent. rating.
120	22 0	1.3	0.0	76.0	560	
123	22.4	0.6	0.0	77.0	510	
120	20.8	1.8	0.0	77.4	555	
120	21.0	1.2	0.0	77.8	560	
131	20.4	3.0	0.0	76.6	565	
131	21.4	3.2	0.0	75.4	585	
131	20.8	1.4	0.0	77.8	570	
124	20.6	3.4	0.0	76.0	565	
124	20.8	3.2	0.0	76.0	580	
131	22.6	2.8	0.0	74.6	520	
131	22.6	1.6	0.0	75.8	540	
131	22.8	0.8	0.0	76.4	550	
131	22.3	1.0	0.0	76.7	548	
121	20.4	3.2	0.0	76.4	530	
94	26.0	0.4	0.0	73.6	530	
94	26.2	0.2	0.0	73.6	535	
122	23.2	0.6	0.0	76.2		
129	20.2	4.8	0.0	75.0		

form CO_2, which runs from 22 to 23 per cent. in spite of the greatly varying gas pressure, *e.g.*, between 6 and 8 a.m. and 10 and 12 p.m. The rear temperature was only 495°F. This is, however, explained by

the fact that the boilers were only running on rated load. The gas in question contained 25 per cent. CO and by referring to Fig. 12 we find that the loss through waste gas was 3.6 per cent. for every 100°F. (stack temperature — boiler room temperature) or a total of 15 per cent., so that the total boiler efficiency, assuming radiation and conduction losses at 5 per cent., is about 80 per cent.

An interesting test on a large load was run by the Pittsburgh Steel

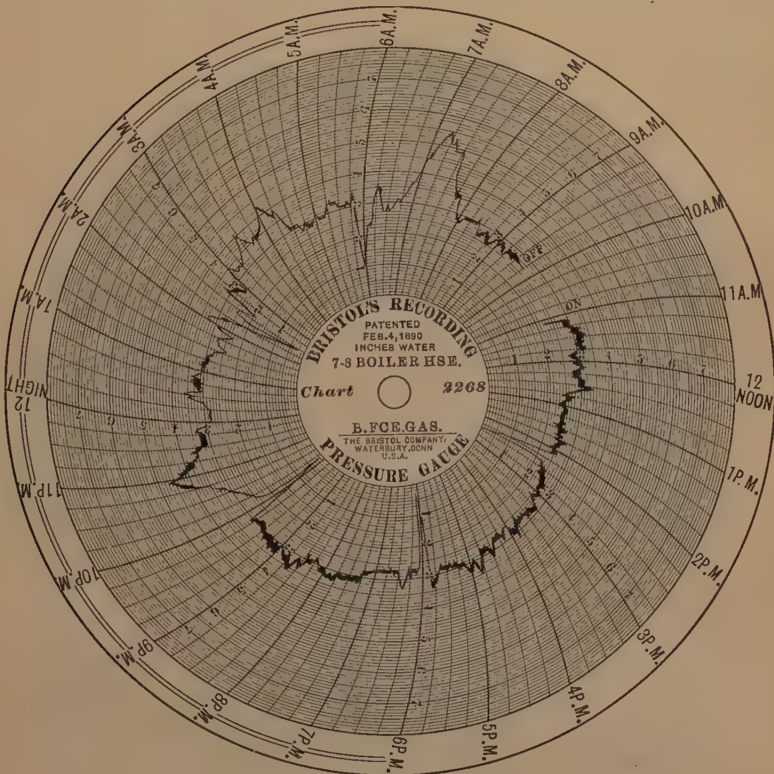

FIG. 16.—GAS-PRESSURE CHART.

Co. at Monessen. I give below the complete results of this test as they were put at my disposal. It must be kept in mind that this test was for the purpose of ascertaining the highest load compatible with good boiler efficiency, as will be seen from the low draft in combustion chamber of only 0.003 in.

The boiler used was equipped with a modification of the Bradshaw burner designed by J. S. Fraser. As I have already explained, it is rather dangerous to run a boiler on its maximum load, since one is constantly verging on incomplete combustion, as was the case in the present instance. Mr. Fraser's modification which is, as will be seen from Fig.

18 the application of the stove-burner principle to the boiler burner, gave nevertheless excellent results. By comparing the analysis in the combustion chamber with the stack analysis it will be seen that there was considerable air infiltration at the back of the boiler, which might of course have been stopped, but for certain reasons this was not done. The total evaporation during 8 hr. was 230,000 lb. water from and at 212°F. which is equivalent to a boiler load of 166 per cent.

The calorific value was stated as 102.8 B.t.u. of cold gas measured at

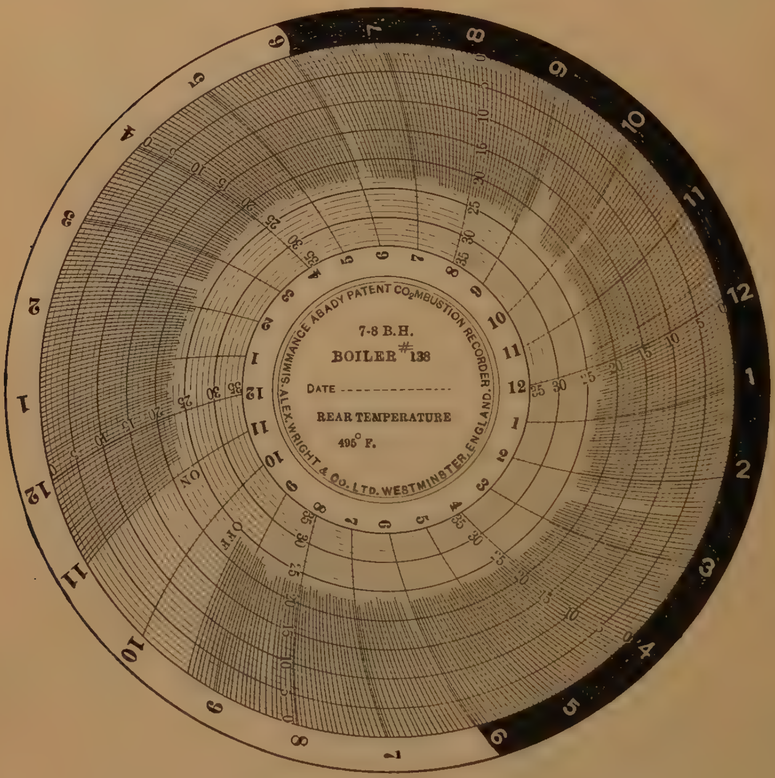

FIG. 17.—CARBON DIOXIDE CHART.

32°F. This is 97.6 B.t.u. measured at 60°F. for cold gas and 105.2 B.t.u. for gas of 377°F. measured at 60°F.

The average stack losses, in accordance with the readings obtained have been, through unburned CO, 1.46 B.t.u.; through sensible heat, 20.32 B.t.u.; total, 21.78 B.t.u. equal to 20.7 per cent.

Assuming the radiation and conduction losses at 5 per cent., the total efficiency is 74.3 per cent., whereas in accordance with Mr. Merwin's figures this efficiency has been 72.3 per cent.

It is interesting to note the astonishingly high combustion tem-

peratures which show that the burner efficiency falls little short of 100 per cent.

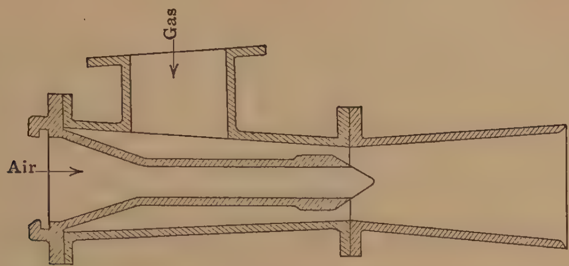

Fig. 18.—Longitudinal Section of Fraser's Modification of Bradshaw Burner.

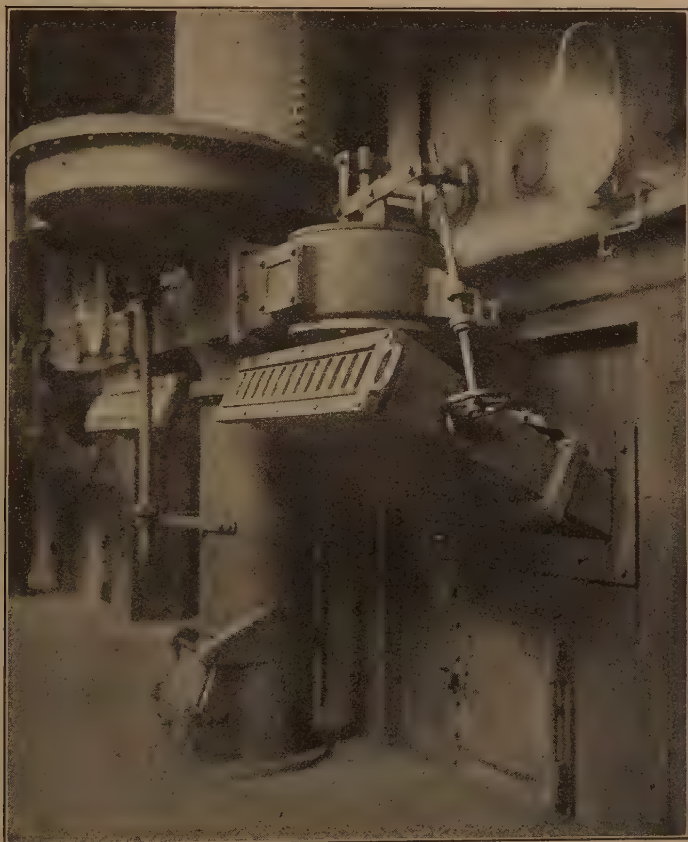

Fig. 19.—Stirling Boiler with Fraser's Modification of Bradshaw Burner.

The importance of the Bradshaw burner as a coal-saving device was shown very clearly by the results obtained at the plants of the Pittsburgh Steel Co. and the Upson Nut Co. during the tests run to prove

Tests Made at Monessen

	Gas Analysis						Rhodes Gas Analysis												Merwin Stack Analysis						
							Combustion Chamber						Stack												
Time	CO_2	CO	H_2	CH_2	N_2	B.t.u.	Gas, Main Pressure	Temp.	Draft	CO	CO_2	O_2	Temp.	CO	CO_2	O_2	Gas, Temp.	Gal. of Water	Time	Temp.	CO	CO_2	O_2	Draft
9:00	13.6	25.2	2.6	0.2	58.4	99.1	17.0	R 2,320 L 2,410	0.0	0.6	24.4	0.9	600	0.2	20.2	4.7	350	....	....	......	0.2	21.0	3.8	.....
9:30	13.1	26.6	...	....	....	.....	16.5	R 2,310 L 2,400	0.0	....	.....	....	590	....	....	....	360	....	9:00	640	0.2	20.9	3.9	0.76
10:00	13.5	26.2	2.8	0.1	57.4	102.2	17.0	R 2,340 L 2,320	0.0	0.8	24.7	0.3	600	0.3	21.0	3.8	340	....	9:30	635	0.2	21.2	3.6	0.74
10:30	....	....	...	....	....	.....	17.0	R 2,390 L 2,420	0.0	....	.....	....	590	....	....	....	340	23,751	10:00	635	0.2	20.6	3.8	0.75
11:00	13.6	26.5	2.8	0.1	57.4	103.3	17.0	R 2,440 L 2,430	0.0	0.5	24.3	0.6	605	0.0	19.9	4.6	380	....	10:30	635	0.4	21.4	3.0	0.75
11:35	....	....	...	....	....	.....	17.5	R 2,380 L 2,380	0.0	....	.....	....	610	....	....	....	410	23,711	11:00	635	0.3	20.4	4.2	0.70
12:00	12.3	26.9	2.7	0.1	58.0	104.3	17.0	R 2,440 L 2,480	0.0	....	.....	....	600	....	....	....	480	....	11:30	640	0.2	20.8	3.8	0.74
12:40	....	....	...	....	....	.....	15.0	....	0.0	....	.....	....	610	....	....	....	320	24,980	12:00	640	0.3	20.4	3.8	0.74
1:00	12.5	26.6	2.7	0.2	58.0	104.4	17.5	R 2,450 L 2,410	0.0	0.8	24.6	0.4	600	0.3	20.9	3.7	400	....	12:30	635	0.0	19.8	4.8	0.72
1:25	....	....	...	....	....	.....	15.0	....	0.0	....	.....	....	610	....	....	....	420	23,751	1:00	630	0.2	20.8	3.8	0.72
2:00	13.2	26.5	2.8	0.1	57.4	103.3	17.5	R 2,470 L 2,440	0.0	0.9	24.7	0.2	600	0.3	21.0	3.8	380	....	1:30	630	0.4	21.2	3.4	0.70
2:80	....	....	...	....	....	.....	16.0	R 2,430 L 2,400	0.0	....	.....	....	600	....	....	....	325	22,482	2:00	635	0.3	20.8	3.8	0.72
3:00	12.5	25.9	2.6	0.2	58.8	101.6	15.0	R 2,400 L 2,390	0.0	0.9	24.3	0.3	600	0.3	20.5	3.9	440	....	2:30	630	0.2	20.6	4.0	0.72
3:30	....	....	...	....	....	.....	17.0	....	0.0	....	.....	....	605	....	....	....	350	22,482	3:00	635	0.3	20.5	4.0	0.73
4:00	12.8	26.4	2.9	0.1	57.8	103.3	16.5	R 2,490 L 2,450	0.0	0.8	24.6	0.4	600	0.3	20.6	4.2	360	....	3:30	650	0.3	20.7	3.8	0.73
4:15	....	....	...	....	....	.....	16.0	R 2,440 L 2,480	0.05	....	.....	....	600	....	....	....	360	23,711	4:00	635	0.3	20.6	3.9	0.71
4:30	....	....	...	....	....	.....	15.5	L 2,500	0.0	....	.....	....	600	....	....	....	380	....	4:30	635	0.3	20.4	4.3	0.71
5:00	11.9	26.4	3.0	0.1	58.6	103.7	16.0	R 2,450 L 2,510	0.0	....	.....	....	600	....	....	....	400	24,113	5:00	......	....	....	....	.....
Averages..	12.8	26.3	2.7	0.13	58.0	102.8	16.5	R 2,408 L 2,435	0.003	0.7	24.5	0.4	601	0.24	20.6	4.1	377	188,981 + 1,864 + 190,845	Total averages	634	0.25	20.6	3.8	0.75
10:00 a.m.	13.1	26.6	per Chemist Rhodes.																					

guarantees. At the Pittsburgh Steel Co.'s plant, the evaporation in the best months previous to the installation of our burners had been 3.6 lb. per pound of coke. For the purpose of proving guarantees a test of four days was run, which showed a maximum evaporation on one day of 4.42 lb. and an average evaporation of 4.25 lb. per pound of coke.

At the Upson Nut Co., previous to the installation of the Bradshaw burner, the average coal consumption over a year had been 1145 tons per month. This coal consumption was reduced to 100 tons per month, a quantity which is actually required for carrying their boilers over the casts. At the same time the Upson Nut Co.'s is now constantly bleeding considerable quantities of gas.

Conclusion

Boiler efficiencies depend on both efficient burners and good boiler "morals." If either of the two is lacking, high efficiency will generally be impossible. Boiler "morals" include proper conditions in the boiler plant itself, as well as intelligent working and close supervision. The regulation of a boiler plant should be done, as far as possible, exclusively on the dampers. With otherwise perfect conditions, efficiencies depend on the boiler loads. With a boiler load of 100 per cent. an efficiency of 80 per cent. is possible. With higher loads, it will soon drop to 75 per cent., but as a rule there will be no need to decrease the efficiency any further. Gas washing is profitable for stoves, but almost always results in losses where it is done for boilers. Moisture in the gas does not seriously affect efficiencies, except in the case of entrained moisture, which should be avoided. Where gas washing is resorted to, the most favorable temperature from the viewpoint of efficiency is 100°F. It is advisable regularly to draw up carbon balance sheets and keep a check of the results obtained by reducing them to pounds of steam per pound of coke.

I beg to acknowledge the assistance kindly given to me, in the preparation of this paper, by J. S. Fraser and Grant D. Bradshaw, and also by Alfred Steinbart, as author of many of the curves.

Discussion

Karl Nibecker, Youngstown, O.—It has been contended that the method of measuring the operation of a blast-furnace boiler plant by means of the amount of steam generated per pound of coke used in the blast furnace is not of any value and even that it is quite misleading.

It has been found, in the plant with which I am connected, that, in general, a blast furnace will operate during a month so close to average conditions that the average monthly measure of the plant efficiency

is indicated by the number of pounds of steam made per pound of coke put into the furnace, neglecting the amount of iron made and the amount of gas used for heating the stoves. For some time, it has been our custom to keep a record of this figure and use it as a general indicator of the operating practice. In comparing this figure from two different plants we have found some difficulty, due to differing conditions. It is conceivable but not altogether probable that a slight difference in burden, kind of ore or coke, would account for this difference. The method of using the gas, whether washed or unwashed, and the plant arrangement, may affect this number. For several plants on which we have worked up this number, we have found that most of the ordinary running irregularities are eliminated during a run of a month or more and that the number is, therefore, satisfactory.

In connection with a plot of the number of pounds of steam generated per pound of coke, we are also plotting the production of the furnaces and the pounds of coke used per ton of production. With these three curves plotted on the same sheet, we believe that the operating man has sufficient information to judge of the efficiency of his plant for any month.

We have checked some of our figures with the figures calculated from the stack gas analyses and temperatures and found that the numbers which we use are quite as satisfactory as those obtained by the more laborious method. Some of the great difficulties with the calculated efficiencies lie in the assumptions which are necessary, such as radiation loss and losses by air infiltration. We feel that it is necessary, wherever there is a considerable number of boilers, such as are used in a plant having three or four blast furnaces, to make a separate and detailed calculation of the efficiency of each boiler and also to measure the steam from each boiler in order to find the desired result.

We do not claim that the figure of pounds of steam per pound of coke is an absolutely reliable and accurate measure of the boiler efficiency but we do believe that it is a very satisfactory indicator. We believe that the calculation of efficiency as found from the fuel and stack gas observations is also not accurate but subject to many errors which in addition to the amount of work involved in calculation does not seem to make it worth while, in spite of the fact that it appears to be a much more scientific method.

On p. 425, the author mentions the importance of running boilers fired with gas and separate boilers fired with coal, when it is necessary to burn coal. We have found that it is possible to obtain very much better economy if some boilers are operated entirely with gas fuel and others operated with coal fuel, than when endeavoring to burn both coal and gas fuel in the same combustion chamber at the same time. We believe that it is possible to burn both coal and gas together but feel

that it requires a great amount of skill and attention in order to avoid burning both of them inefficiently.

On p. 424, the author advocates the use of hot, unwashed gas under boilers as the sensible heat is thus conserved. We of course appreciate that washed gas eliminates the accumulation of dirt on the boiler tubes, which is a source of great loss of boiler capacity and efficiency. We have conducted some tests where it was possible to note a considerable drop in capacity and rise in stack temperature between the cleaning periods of the boiler.

If mechanical soot blowers are properly installed and operated frequently, we feel that the efficiency can be maintained at a satisfactorily high average even when using unwashed gas. With the mechanical soot blowers installed, I believe a more economical plant is obtained when using unwashed gas than by the use of washed gas.

S. K. Varnes, Steelton, Pa.—I consider Mr. Huessener's paper very able and wish to emphasize his advocacy of keeping accurate operating records. We have been keeping operating records for practically all of the four years since we installed feed-water meters and have found these records invaluable. We keep segregated costs of the different operations. We have adopted as our unit of calculation 1,000 lb. of water evaporated, because this requires no conversion from our meter readings and enables us to compare not only our boiler efficiency from month to month but our repair cost items as well. We also keep a chart on which is plotted the monthly average boiler horsepower obtained per ton of pig iron per day. This basis of measuring efficiency is different from the one advocated by Mr. Huessener, and I am inclined to think that his is better, because alongside of the first curve we have to plot a second showing the fuel ratio in the furnaces, in order to make the first curve really intelligible. If we were starting anew we should probably adopt Mr. Huessener's suggestion.

I might say, for the benefit of those who have not studied the problem as deeply as we have, that most of the trouble we have found with air infiltration is with infiltration through cracked brickwork and through joints between the brickwork and steelwork. If the cracks and joints are carefully cared for, the leakage through a well-built wall is almost negligible.

We also have found it very profitable, when working with ores producing a dirty gas, to blow the flues once in every 12 hr. We have a number of stack thermometer charts which are practically duplicates of those shown in Mr. Huessener's paper, in their general form. We have obtained rather satisfactory results in combustion efficiency by sending a man into the boiler house with an Orsat gas-analysis apparatus and making adjustments on the boiler burner, keeping the foreman of the boiler house informed of the results obtained. We find that after a few

weeks' experience a boiler foreman is able to judge the combustion from the appearance of the flame, but his efficiency in this respect falls off rather rapidly and it is necessary to repeat these instructions at intervals of six to eight weeks.

We have been operating two of the Bradshaw burners for several weeks on one 500-hp. boiler, making comparative tests with the burners we were using formerly, which were of the common type of Bunsen burner in use in blast-furnace plants. We find that, although we can, with great care, get ideal combustion conditions with the old burners and maintain good efficiency, the adjustments are much easier to make with the Bradshaw burner and the operating advantages of the latter are important even though the actual efficiency obtained by us was not very much greater than that of the old burners. The burner which automatically regulates the air admission and thereby maintains high combustion efficiency will certainly show considerable advantage over a burner which has to be constantly watched and adjusted with each variation in gas pressure and stack draft, which, in turn, is largely dependent on the gas pressure in the main.

J. W. Richards, So. Bethlehem, Pa.—I think this paper is, at least as far as boilers are concerned, a plea for the dry cleaning of gas instead of wet cleaning, because the dry cleaning saves the sensible heat of the gas. The statement that the best measure of efficiency is the amount of steam obtained per pound of coke put into the furnace may be misunderstood. This can only be a measure of the efficiency of the boilers on the assumption that the furnace is making gas of regular composition; if not, then it is just as much a measure of the inefficiency of the utilization of the coke in the blast furnace.

K. Huessener, Pittsburgh, Pa.—I quite agree with Professor Richards that the blast furnace is no gas producer, and that, therefore, in using the pounds of steam produced per pound of coke in the furnace as the measure of efficiency, the varying conditions have to be taken into account.

It may interest you to know that the Cambria Steel Co. has been recording these figures for over two years in connection with very close supervision of the boilers. They have found that the amount of steam raised per pound of coke put into the furnace is influenced very little in the long run by the irregularities of the furnace.

Manganese-Steel Castings in the Mining Industry

BY WALTER S. McKEE,* CHICAGO, ILL.

(New York Meeting, February, 1916)

WHEN mixed with common cast steel in quantities ranging from 11 to 13½ per cent. and properly treated, manganese increases the ductility of the metal and adds greatly to its toughness and resistance to abrasive action, which properties comprise the distinctive characteristics that make manganese steel so valuable for many purposes at the present time.

At present the manganese-steel foundries of this country have an annual capacity of about 60,000 tons and most of the growth has occurred within the past 10 years. In the early days of the industry it was impracticable to make manganese castings weighing over a few thousand pounds, but at present crusher heads, rolling-mill pinions, and similar castings weighing up to 30,000 lb. each are produced, and it does not appear that the limit of weight has been reached by a considerable margin.

Manganese steel is similar in analysis to ordinary first-class converter metal except that it is high in carbon and contains from 11 to 13.5 per cent. of manganese. From one standpoint manganese steel is similar to malleable iron in that the casting as it comes out of the sand is hard and brittle and must be made ductile by a heat-treating process. The heat treatment is an essential part of the process of making manganese-steel castings and must be properly carried out to secure satisfactory results. The necessity for heat treatment limits the thickness of sections which may be made of manganese steel. Originally the annealing depth was comparatively small, but researches carried on during the past few years have resulted in an increase in the thickness of section up to 5½ in., which it is possible to treat successfully. Large castings are cored out to a minimum thickness of 5½ in., thus not only eliminating some of the internal stresses which are invariably set up in a very thick casting, but also reducing the weight and inertia of the piece, an important consideration if the casting is to be a moving part in a machine. At the same time, the thickness of 5½ in. is sufficient to permit the use of manganese steel for the heaviest classes of machinery.

* Vice President, American Manganese Steel Co.

Before taking up the application of manganese-steel castings to machinery used in the mining industry, a few details covering the chemical and physical characteristics as well as the micro-structure of this material will be presented. Chemically, cast manganese steel has about the following composition:

	Per Cent.
Carbon	1.25
Silicon	0.30
Manganese	12.50
Sulphur, less than	0.02
Phosphorus, about	0.08

The following results are an average of 19 physical tests of manganese steel: Elastic limit, 53,396 lb. per square inch; tensile strength, 108,460 lb. per square inch; elongation in 2 in., 33.71 per cent.; reduction of area, 38.56 per cent.

The distinctive characteristics of manganese steel are clearly indicated by the microscope. As is well known, manganese is present in all ordinary steel, but the metal does not become austenitic until about 6 per cent. manganese and 0.80 per cent. carbon are introduced. Commercial austenitic manganese steel generally contains from 11 to 13.5 per cent. manganese and from 1.00 to 1.30 per cent. carbon.

In the cast state, manganese steel is composed principally of austenite and free cementite, austenite being a solution of iron, manganese, and the carbide of iron and manganese, while free cementite is composed of the carbides of iron and manganese which remain undissolved. Free cementite is hard and brittle, therefore manganese steel is also brittle in the cast condition. We thus have a metallographic explanation of the chief characteristics of manganese steel. Microscopic examination reveals the fact that if the steel is heated to a proper temperature and quenched in water, the free cementite is dissolved and the whole becomes austenite.

The accompanying photomicrographs, Figs. 1 to 5, show the structures of manganese steel as cast and after treatment. Fig. 1 shows the structure of manganese steel in the cast condition. Fig. 2 shows the structure of the same steel heated to a temperature of 1,750°F. and quenched in water. The heat treatment was not correct, and although much of the free cementite has disappeared, some of it still remains in chunks as shown by the dark spots. Fig. 3 shows the structure of the same steel heated to a temperature of 1,800°F. and quenched in water. In this case the heat treatment is correct and all of the free cementite has disappeared, leaving a purely austenitic structure.

The effect of subsequent heating and cooling in air is shown in Figs. 4 and 5. Fig. 4 shows the structure of a manganese steel which had received a proper heat treatment and was afterward heated to a

temperature of 700°F. and allowed to cool in air. The structure is exactly similar to that shown in Fig. 3; the steel, therefore, has not been injured. Fig. 5 shows the structure of the same steel heated to

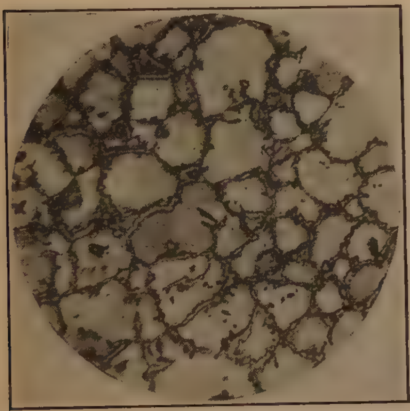

Fig. 1.—Structure of Manganese Steel in Cast Condition. × 80 Diameters.

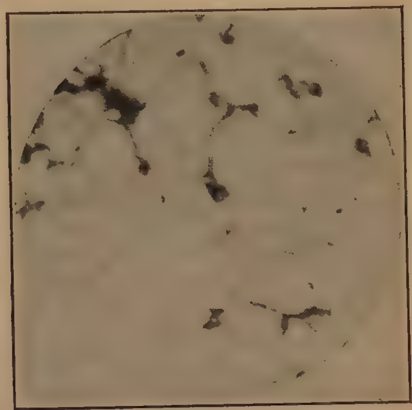

Fig. 2.—The Same Steel Incorrectly Heat Treated, Some Cementite Visible.

a temperature of 750°F. and allowed to cool in air. A heavy separation of cementite, shown by the needle-like particles, is clearly visible. This proves that heating the steel to over 700°F. destroys the purely austenitic

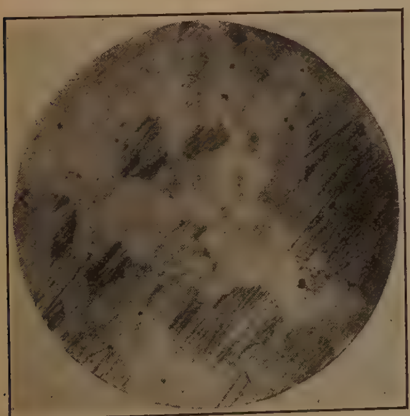

Fig. 3.—The Same Steel Properly Heat Treated, Resulting in a Pure Austenitic Structure. × 80 Diameters.

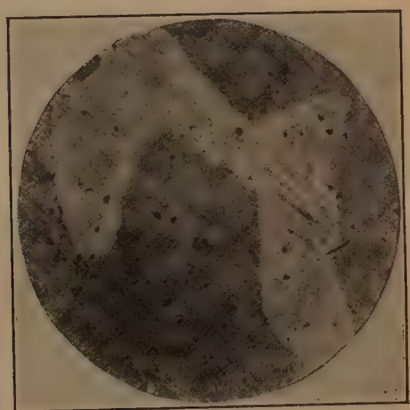

Fig. 4.—Manganese Steel Correctly Heat Treated and Subsequently Heated to a Temperature of 700°F. and Cooled in Air Without Changing the Austenitic Structure. × 80 Diameters.

structure, seriously injuring the quality of the metal. It is therefore imperative that manganese steel should not be heated to a temperature exceeding 700°F. after being heat treated. The experiments on which

the foregoing photomicrographs are based were carried out with the aid of an electric furnace equipped with a pyrometer.

By the Brinell test, manganese steel toughened or untoughened shows the moderate average hardness number of about 200. The extreme outer surface of the treated steel shows a slightly lower number than at a depth of about ⅛ in., from which point the number remains constant to the core of the casting. The lowering of the hardness number at the surface is due to the oxidization of the carbon during the heat treatment.

The qualities of manganese steel have led to its adoption extensively for the vital parts of heavy-duty machinery, particularly parts subjected to unusual wear, shock, etc. Manganese-steel castings are used today in the construction of wearing parts for the following kinds of machinery: Rock and ore crushers, coal breakers, ball mills, tube mills, pulverizers,

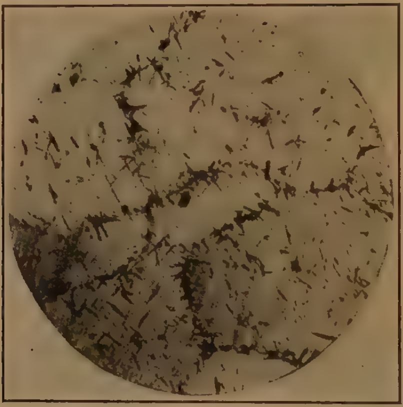

Fig. 5.—The Same Steel Heated to 750°F. and Allowed to Cool in Air Resulting in the Precipitation of Cementite Injuring the Quality of the Metal. × 80 Diameters.

comminuters, clay mills, traveling cranes, gold dredges, harbor dredges, steam shovels, ditching machines, pig-casting machines, pug mills, coke machines, steel-rolling machinery, stamp mills, cable haulage systems, screening apparatus, centrifugal sand pumps, grab buckets, concrete mixers, etc. Manganese steel is also extensively used in the manufacture of steam and electric railway track work, elevator buckets, gears, pinions, mine and skip car wheels, and for numerous other purposes.

The use of manganese steel for tube-mill liners is a recent development which is of special interest at this time because of the curtailment in the supply of lining materials previously used, by reason of the war abroad. Until the introduction of manganese-steel liners, tube mills were lined almost exclusively with silex blocks imported from Europe, principally from Denmark. The pebbles used in tube mills also have been imported, chiefly from France and Denmark. The manganese-steel liners have been developed to the point where they form a satis-

factory substitute for silex blocks, and experiments are now being conducted on the use of 2- and 3-in. diameter manganese-steel balls in place of imported pebbles. A manganese-steel tube-mill liner known as the Komata, developed by F. C. Brown, is now used with good results. As indicated in Fig. 6, it consists simply of a series of plates and lifting bars securely attached to the shell of the mill by square-headed taper bolts, drawn up from the outside, so that the plates are held in place even when worn extremely thin. The lifting bars perform an important

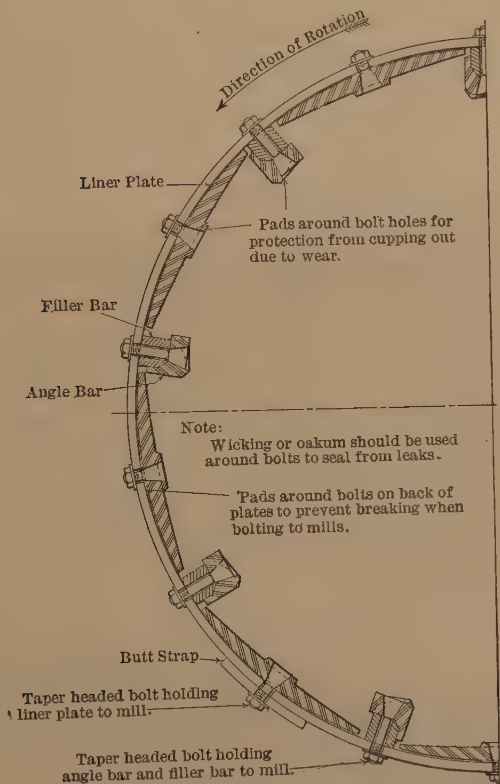

Fig. 6.—Half-Section of a Tube Mill Fitted With Manganese-Steel Komata Lining Plates and Lifting Bars.

function, their action being similar to the bars installed in Bradford coal breakers. The bars carry the pebbles up to a point considerably beyond the center line of the mill, causing an almost perfect cascading action. For this reason the pebbles maintain a spherical shape and the great commotion and rolling action of the pebbles produces an effective grinding action. The pebbles do not slip on the liners, with the result that the life of the plates is prolonged and the tendency of pebbles to become flat is prevented.

The lifting bars are invariably made of manganese steel because of the great wear to which they are subjected. The lining plates are preferably made of the same material, but if local circumstances warrant, they can also be cast of less expensive semi-steel or white iron at local foundries. To prevent the liner from cupping around the bolt holes, raised bosses are placed on the castings around each hole.

In operation the pulp soon gets behind the plates and forms a cushion or backing. The ribs, or lifting bars, are spaced from 18 to 20 in. apart according to the diameter of the mill. The filler bars which are placed underneath the lifting bars are not exposed to wear and therefore are made of soft cast iron. The joints in the bars are staggered with the joints of the liner plates to prevent the formation of grooves or channels in which the pulp might travel and wear the shell of the mill.

When a manganese-steel liner is employed, the speed of the mill has an important influence on the results obtained. The correct speed can be derived from the following formula:

$$\text{Revolutions per minute} = 183 \div \sqrt{D}$$

where D is the diameter of the mill in inches.

To secure satisfactory results, the pebble load should be about 3 to 5 in. below the center line of the mill, depending on the diameter. When grinding ordinary coarsely crushed battery pulp, the new pebbles fed into the mill should not exceed 2½ in. or, in any case, 3 in. in diameter.

For the year ended June 30, 1914, one of the largest milling companies in Nevada reported the following consumption of lining material: Manganese-steel lifting bars, 16,633 lb.; semi-steel liner plates, 49,916 lb. During this period, 341,354 tons of ore were ground, comprising the discharge of 12 batteries equipped with 3-mesh screens, together with the discharge from eight batteries passed through Chilean mills equipped with 16-mesh screens, both products being passed through Dorr classifiers before the sands were fed to six 5 by 22 ft. tube mills. The actual sliming done by the tube mills was estimated at 100 tons per day per mill, 80 per cent. of the product passing through a 200-mesh screen. The pebble consumption for a similar period was 1,647,524 lb.

At this plant a set of manganese-steel lifting bars lasts about 14 months and a set of semi-steel liner plates, which are 1 11/16 in. thick at the center, tapering to 11/16 in. at the edges, lasts about 23 months.

In some of the mills manganese-steel liner plates also were used. They were in service 29 months and when removed weighed less than 30 lb. each. These plates were 1⅛ in. thick at the center, tapering to ⅜ in. in thickness at the edge.

The consumption of metal and pebbles per ton of material ground, according to the figures given previously, was as follows:

	Consumption per Ton of Material Ground, Pounds
Manganese-steel lifting bars	0.049
Semi-steel liner plates	0.137
Pebbles	4.83

In the mills in which the foregoing tests were made, it is not customary to renew the whole liner at once, but to remove worn-out pieces from time to time as necessity demands.

Some interesting tests of manganese-steel liners have also been made on a Schmidt-Davidson type mill in operation in New Zealand. This mill is 4 ft. 6 in. in diameter and is run at 23 r.p.m., which was found to be the most effective speed when using lifting bars. The liners in this case were of hard, white cast iron made at a local foundry. When new, the liners were 1 in. thick in the center, tapering to ⅜ in. at the edges, the lining weighing 5,940 lb. There were 10 rows of ribs in the mill with an aggregate weight of 2,347 lb. The liners were used for 75½ weeks and the angle bars for 60½ weeks before they required replacing. From 187 to 232 tons of ore were passed through the mill daily, these figures including the coarse sand which was returned to the mill as many times as necessary. The stamp delivered an average of 76 tons of ore to the mill daily. The pebbles were from 1½ to 2½ in. in diameter.

During the last six months of the run, the fineness of grinding was increased and for this reason the quantity of ore fed to the mill from the stamps was reduced to 55 tons per day, exclusive of returns. The fineness of the ore before and after grinding, when crushing this tonnage, averaged as follows:

Screen Mesh	Battery Discharge, Per Cent.	Tube Mill Discharge Per Cent.
+10	6.2	.....
+20	6.7	.....
+40	20.2	.....
+60	8.3	.....
+80	7.0	.....
+100	5.6	0.6
+150	10.4	6.6
+200	2.4	4.7
−200	33.2	88.1
	100.0	100.0

The pulp passing through the tube mill varied between 1 and 1.4 parts of moisture to 1 of ore by weight. The adoption of this special form of liner is said to have reduced the consumption of pebbles by 70 per cent., the liner cost having been cut 37 per cent.

A company at Tonopah, Nev., is operating two 5 by 18 ft. tube mills fitted with Komata linings. These mills run at 24.4 r.p.m. and they re-

quire 98.67 hp., or 46.335 hp. per mill. The mills are fed with the discharge from stamps which are equipped chiefly with 6-mesh screens, a few batteries being provided with 4-mesh screens. The feed contains 38 per cent. moisture. With a pebble consumption of approximately 4 lb. per ton of ore ground, the results are as follows:

Screen Mesh	Battery Discharge Per Cent.	Tube Mill Discharge, Per Cent.	Finished Product Per Cent.
+10	10.8	....	....
+14	10.7	....	....
+20	10.0	....	....
+28	10.3	....	....
+35	7.7	6.0	....
+48	7.1	2.4	....
+65	5.6	5.3	....
+100	....	16.1	22.0
+110	6.0	....	....
+150	6.6	28.3	12.2
+200	4.0	14.2	7.3
−200	21.2	33.1	78.3

Manganese steel has been found particularly suitable for dredge buckets in the gold-dredging districts of Montana, California and elsewhere, the life of the manganese-steel buckets being reckoned by years instead of months. The buckets usually are cast solid and the lips riveted in place; they range from 600 to 5,000 lb. in weight each, the average being about 2,200 lb. A number of buckets of this type have been in service since September, 1909, and judging from present indications their total life will be from 10 to 12 years.

Usually placer dredge buckets are removed from service because the metal underneath the back eye wears away. In some cases the buckets are worn entirely through to the bushing at this point before they are taken from the line. The metal underneath the back eye ranges from 1½ to 3 in. in thickness, depending on the size of the bucket. Some observations were recently made of the wear on 50 manganese-steel buckets placed in service on a dredge in Montana in December, 1911. These buckets have a capacity of 9 cu. ft. each. When new, the metal underneath the back eye had an average thickness of 2½ in. After having been in service for three years, actual running time, the buckets were again measured and the metal under the back eye was found to have been worn to an average thickness of 1⅞ in. In other words, about ⅝ in. of metal was worn away. This means that 1¼ in. of metal are still available for wear, assuming that the buckets are worn out when the thickness is reduced to ⅝ in. Since the wear has amounted to 5/24 in. per year, the estimated total life of the buckets is nine years.

Another set of measurements was made recently on a line of 86 5-cu. ft. dredge buckets which have been in service in Colorado for 32 months, actual running time, covering the period from March, 1911, to Aug.

5, 1915. These buckets which when new had a thickness of 2⅛ in. underneath the back eye showed an average wear for the period of ⅜ in. Assuming that the 5-cu. ft. manganese-steel buckets would be discarded when but ⅝ in. of metal is left underneath the back eye, the estimated life is 10½ years running time. It is estimated that the average life of a carbon-steel bucket line on this particular dredge would not exceed 30 months.

Most dredge buckets now made are of the two-piece type, the bottom and hood being cast integrally. With this design of bucket, leakage is avoided and there is no danger of loss of valuable material through the rivet holes while the bucket is carrying its load from pond to the upper tumbler. A short lip can be used on the two-piece buckets and comparatively few rivets are required.

The tumblers on gold dredges also are frequently made of cast manganese steel, some of them weighing 24,578 lb. The tumbler is simple in construction, consisting of nothing more than a spool cast in one piece, the bore being finished by grinding. Renewable wearing plates are eliminated on both the inside and outside of the flanges and on the barrel. Although the round tumbler is used successfully with the ordinary type of bucket, it is considered preferable to widen the bottom of the bucket somewhat when a tumbler of this type is employed. This usually is accomplished by placing a flange along the outside of the bucket at the bottom between the front and rear eyes.

Screen plates are also being successfully made from manganese steel. When used in connection with gold dredges, the hole may be either ½ in. in diameter, tapering to ⅝ in. on the outside, or ⅝ in. in diameter, tapering to ¾ in. on the outside. In order to overcome the difficulties of casting round holes in manganese steel, some plates have been designed with rectangular slots about ½ in. or ⅝ in. wide on the inside, tapering to ⅝ or ¾ in. on the outside, the slots being 1¼ in. long and set diagonally. Elevator buckets of the same thickness as malleable iron buckets are now made of manganese steel, the latter giving six times the service of the malleable iron.

The application of manganese steel to steam-shovel dippers was one of the first problems attacked by the manufacturers of this material in this country, the wear in this service being particularly severe. A dipper known as the Missabe type is made entirely of manganese steel. Its body consists of but two castings, the front and back halves, which give maximum stiffness and rigidity. This design permits the construction of a large dipper with only a few rivets. The bail brackets are set at an angle conforming to the line of pull on the bail. They are butted against shoulders or offsets which are formed integrally with the sides of the front castings and are riveted to the front half of the dipper, the joints between the front and back halves being placed at the rear of the brackets, thus

throwing all the digging strains on the front casting, and relieving the rivets at the joints from shearing stresses. The teeth, which are easily renewable, are of cast manganese steel.

Dippers of the Missabe type of 10-cu. yd. capacity, which are believed to be the largest manganese-steel dippers ever made, were used on dredges working in the Culebra Cut, Panama Canal. The weight of the empty

FIG. 7.—FIVE CUBIC YARD MISSABE TYPE MANGANESE-STEEL DIPPER WITH VANDERHOEF FRONT.

dipper is 37,600 lb. and its overall dimensions are 9 by 9 by 10½ ft. The lips are 3¼ in. thick and the fronts are the same thickness underneath the teeth. Between the teeth the fronts are 1½ in. thick, the thickness at the bottom band being 3¼ in. The backs are 1¼ in. thick, except at the bottom where the thickness is increased to 3¼ in.

Another improved dipper front is known as the "Vanderhoef," in which the front and the dipper-tooth bases are cast integrally. The renewable points are substantially designed and are of the reversible

type. Among the advantages of this design may be mentioned the elimination of rivets, correct digging angle, increased room inside the dipper, due to the position of the base portions, great strength coupled with rigidity of the entire front, and economy in repairs and renewals. Fig. 7 shows in detail a Missabe type manganese-steel dipper of 5-cu. yd. capacity, equipped with a Vanderhoef front.

Fig. 8 shows a cast manganese-steel ventilated head of the Black type, suitable for a No. 21 Gates gyratory crusher. This casting weighs 21,360 lb. As shown in the illustration, it is provided with a series of

Fig. 8.—Black Type Ventilated Head for No. 21 Gates Crusher.

vertical flues or cores running from top to bottom which permit a circulation of air between the wearing surface and the shaft. This tends to prevent the transmission, from the wearing surface to the shaft, of the heat incident to crushing. As compared with a solid head, the weight is reduced from 20 to 30 per cent.

Manganese steel, although not suitable for railway car wheels on account of its comparative softness, has been found satisfactory for mine- and skip-car and crane wheels. For crane wheels carrying heavy rolling loads, manganese steel has been found particularly satisfactory. Manganese-steel crane wheels are in service today carrying loads of 90,000 lb. each, although 30,000 lb. has always been considered the limit of safe working

load for chilled-iron wheels. A microscopic examination of a manganese-steel wheel under load will show a flat spot instead of a line contact as in the case of hard-metal wheels; manganese steel under these conditions is springy and recoils as the load is released. The wheels also exhibit strong resistance to flange wear. This feature is worthy of consideration in view of the fact that derailments on curves are usually caused by the flanges having worn to the angle of least frictional resistance, allowing the inherent straight line force to push the wheel flange over the rail.

Where the loads are heavy, manganese steel is now being extensively employed for gears. For the past two years manganese-steel spur gears have been made with the teeth ground mechanically true, up to 44-in. pitch diameter with a 10-in. width of face. Many ordinary steel gears are cut with a range cutter which does not leave the rolling contact theoretically correct, whereas in grinding manganese-steel gears, a trimming arrangement is placed on the grinder for the purpose of dressing the emery wheel which shapes the tooth to the proper contour.

After two years' continuous service, the wear on a set of heavy high-speed manganese-steel nest gears was recently measured. These gears had a 10-in. face and when new the thickness of the tooth at the pitch line was 0.684 in. At the conclusion of the two-year period, the tooth measured 0.601 in., the wear being 0.083 in. These gears are still in service after having more than earned their original cost, and barring accident they will be good for many years to come.

Manganese steel is now employed extensively in rolling-mill machinery; it has been found particularly suitable for pinions on heavy roughing and blooming mills. Other parts around steel mills now being made of manganese steel are pipe-drawing balls, chafing plates, sprockets, cams, draw chains, unloader chains, conveyor chains, buckets, screens, pulverizer hammers, dry and wet pan parts, crusher castings, coupling boxes, spindles, and backing rolls.

For the production of manganese-steel castings the shop practice is fundamentally similar to that of an ordinary steel foundry, but a number of factors not vital in ordinary work must be considered. Among other points, the shrinkage is 5/16 in. per foot as against 3/16 to 1/4 in. for ordinary steel. For this reason the founders frequently prefer to make their own patterns from drawings supplied by their customers. Where patterns are furnished, sometimes they have to be modified or rebuilt.

The distribution of the metal in the casting is also important, since the heat treatment and cooling produce stresses which must be taken care of. At times it is necessary to add metal to a casting temporarily to withstand shrinkage strains, such additions of metal being removed in the machine shop. Manganese steel, being comparatively expensive, must be used economically, for which reason designs or patterns occasionally have to be altered.

In making manganese steel, side-blow converters are commonly employed and the 80 per cent. ferromanganese with which the steel is alloyed is melted in separate crucible furnaces. The melted alloy is placed in the bottom of the ladle and the converter metal is poured in on the top.

Manganese acts as a strong scavenger, removing the gases and impurities from the metal, leaving it condensed and homogeneous. Before the molds are poured, the ladles are allowed to stand for a time to permit the alloy to perform its function, after which the accumulation of slag is removed.

The annealing process is particularly important. The temperatures and the time the work is allowed to remain in the furnace vary considerably, depending upon the nature of the piece; the temperature ranges from 1,600 to 2,200°F. and the time of annealing from 4 to 24 hr.

The gates and risers of castings can be broken off as in malleable foundries. The cores are removed with pneumatic drills and the exterior surfaces are cleaned and trimmed by grinding.

Castings which are to be accurate are finished in the machine shop by grinding, much of the work requiring special grinding equipment. Holes more than 1¼ in. in diameter are cored out of the casting and ground to size. When it is necessary to drill smaller holes or to cut threads, soft steel or wrought-iron inserts are set in the molds at the desired points, like chaplets, and the metal is cast around them. Except in small holes, inserts or bushings are not necessary where keyways are cut, since these can now be ground successfully. Sometimes bushings are set in the hubs of wheels to be machined by ordinary methods.

The foreman continually exercises his judgment as to the wheels and cuts to be employed; coarse wheels and heavy cuts are used for removing stock rapidly, though it is never accomplished as easily as would be the case were it possible to use ordinary cutting tools.

A complete manganese-steel castings plant, therefore, includes a steel foundry with special equipment for melting ferromanganese and annealing castings, a pattern shop, and a machine shop equipped with heavy-duty standard and special grinding apparatus.

Discussion

J. W. Richards, So. Bethlehem, Pa.—I regret that the author has overlooked the great usefulness of the electric furnace for the melting of ferromanganese in the manufacture of manganese steel.

Walter S. McKee, Chicago, Ill. (communication to the Secretary*).—In an article as brief as mine it was not possible to fully cover the metallurgical field; this accounts for my not mentioning the electric furnace for the melting of ferromanganese in the manufacture of manganese steel, although the advantages of such an apparatus are fully appreciated by me.

* Received Mar. 2, 1916.

Frederick Laist,* Anaconda, Mont. (communication to the Secretary†).—Manganese steel has proved very serviceable in our plant. We are using manganese steel for the jaw plates on the 5- by 15-in. crushers in the concentrator instead of cast-iron jaw plates which we formerly used. The manganese steel wears five times as long as the cast iron. We are also using manganese steel in our 8- by 20-in. crushers. In our primary crushers, which are 12 by 24 in., we still use cast iron because the manganese-steel plates wore unevenly and left shoulders which caused trouble in setting up the crusher. Another reason for preferring cast iron in these crushers is that they grip the ore better on account of wearing of the rougher surface. The surface of the manganese steel becomes polished and large pieces of ore are not drawn in.

We recently tested 8- by 16-in. manganese-steel elevator buckets working on 2-in. undersize against malleable-iron buckets and found that manganese-steel buckets lasted two and one-half times as long as the malleable-iron buckets.

We use manganese-steel screen plates on our 2-in. trommels and have found that they last about five times as long as cast iron.

We also used manganese-steel wheels and rails under the 20-deck round table in our slimes-treatment plant. We first used cast-iron rails and wheels and found that these wore badly at the end of two or three weeks, while the life of the manganese steel seemed to be almost indefinite; we never had occasion to change any of the manganese steel rail sections or wheels in the year and nine months that the plant was in operation.

Albert E. Wiggin,‡ Anaconda, Mont. (communication to the Secretary§).—In regard to the use of manganese steel for grinding mill linings, I would say that we are at present using the Cascade type of lining made of manganese steel. This lining was designed, I think, by the American Manganese Steel Co. We have not used this lining long enough to form any definite opinion of its efficiency, but, judging from the effects of wear on one of these linings which has been in service about two months, we estimate that the life of the lining will be at least a year.

We have not tried the Komata type of lining, but we have this in mind, and intend to try it very soon. We are inclined to think that the Komata type of lining may give us too much lifting action. It seems to us that for fine grinding, very little lifting action is required, only a sufficient amount to keep the balls in motion. However, we may be wrong in this, and only a trial of this type of lining will prove this.

* Metallurgical Manager, Anaconda Copper Mining Co.

† Received Feb. 16, 1916.

‡ Superintendent of Concentration, Anaconda Copper Mining Co.

§ Received Feb. 18, 1916.

A Chemical Explanation of the Effect of Oxygen in Strengthening Cast Iron

BY W. McA. JOHNSON, HARTFORD, CONN.

(New York Meeting, February, 1916)

THE work of J. E. Johnson, Jr., on the effect of small amounts of oxygen in cast iron in increasing its strength and resistance to shock, is of interest from the technical and scientific standpoints. The following exposition of the theory carries Mr. Johnson's explanation further and, in my opinion, will disclose certain related phenomena and perhaps shed further light on the subject.

Some facts, so novel in character as to excite incredulity, have been accepted finally by many metallurgical authorities. One of these is that oxygen to the amount of 0.060 per cent. in cast iron gives it a breaking strength of 3,500 lb. per square inch, compared with a breaking strength of 2,500 lb. per square inch in cast iron of the same composition with respect to elements other than oxygen, but having only 0.010 per cent. oxygen. The "oxygenated" cast iron, even if made in a coke furnace, has properties equivalent to the best charcoal pig iron. Mr. Johnson has also established the fact that the variations in combined carbon cannot account for the results. By microphotographic evidence he has shown that in oxygenated cast iron the graphite particles are dense and spherical, while in well-deoxidized cast iron the graphite particles are flaky and leaf-like. It is just a matter of common sense that iron of the latter structure is much weaker than the former. The analytical and microscopical work has been done with care and there is no doubt in my mind about the accuracy of the experimental results. In this paper then, we shall consider it as accepted that small amounts (0.060 per cent.) of oxygen increase the strength of cast iron by producing a structure in which the round and solid particles of graphite are surrounded firmly by a principal matrix of ferrite.

The purpose of this paper is to show that the reason why the graphite particles are round is founded on the fact that the particles of any precipitate are made denser and harder by the presence of a reagent having a solvent action on the precipitate. This can be stated as a law, although I have never seen it so given in any textbook.

Let us then, for purposes of illustration, give a concrete example and develop the theory of such mechanical and chemical actions as occur:

It is well known that, by a soluble sulphate, a barium salt in solution is precipitated in a dense and easily filtered body when the solution is

strongly acid with hydrochloric acid, while in a neutral solution barium sulphate is thrown down as a flocculent body. The same is true of silver chloride precipitated out of a silver nitrate solution by hydrochloric acid with nitric acid as the excess reagent. If we increase the proportion of hydrochloric acid in the case of barium sulphate, and the nitric acid in the case of silver chloride, the precipitate is thereby made dense and easily filtered. In any case, the mechanism of the operation is the same, provided a sufficient interval of time be given to allow the mechanistic operation to proceed to equilibrium.

It may be remembered that this equilibrium is not a static, but rather a kinetic one. By this it is meant that an interchange of molecules from the solid to the solution is continuously taking place. We have the velocities of the two reactions that take place as equal and opposite.

We have, as is well known, the two equal and opposite reactions:

1. $AgNO_3 + HCl = AgCl + HNO_3$
   Velocity $= V_1$ Precipitation
2. $HNO_3 + AgCl = AgNO_3 + HCl$
   Velocity $= V_2$. Dissolution

To use an old-fashioned term, the influence of a solvent on a precipitate is as follows:

The first form of any precipitate is particles of molecular fineness. If these particles subsequently agglomerate, they agglomerate in long dendritic forms, in accordance with the probable effect of a weak force of cohesion acting on an infinite number of solid particles. In general, the velocity indicated above as V_1 is less than V_2, and then the precipitate would be dissolved; on the contrary, whenever V_1 is greater than V_2, precipitation would take place.

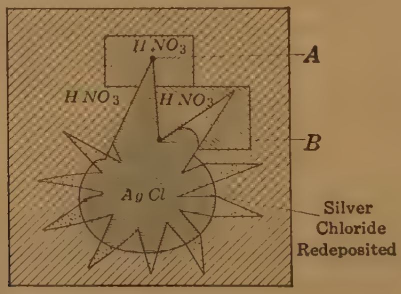

FIG. 1.

Considering now a case when V_1 is greater than V_2 and precipitation is taking place, and a particle having a body terminating in a point: It is obvious that the instantaneous effect of V_1 would be less on the point of such a particle than it would be about the center, on account of the law of mass action. Therefore, the particle would tend to be dissolved

at the point and would then tend to be reprecipitated near the center of cohesive force of each particle. These interchanges would be comparatively rapid at first and then slower, but the general effect would be that the precipitate would become more and more composed of particles of such shape and density that the force of cohesion would act more strongly as the molecules came into closer mutual relation.

In Fig. 1, at the point A, silver chloride is almost all surrounded by nitric acid. At the point B, it is almost all surrounded by silver chloride. In accordance then with the law of mass-action, the chances will be to dissolve at the point A and reprecipitate at the point B. This is an interchange that is taking place at first with a bombardment effect. It is simply a capture and recapture of atoms and molecules until a strong molecular citadel is made. Speaking in military metaphor it is not unlike what is now happening at Verdun, where the exposed positions are being taken by the enemy, with a resulting increase in the strength of the consolidated position.

The net effect of this would be that the particles of spherical shape would increase in size and density according as the magnitudes V_1 and V_2 were unequal instantaneously, or, explaining it in another way, according to the integrated heterogeneity of V_1 and V_2.

Just this action occurs with oxygenated cast iron; we have the solvent action of oxygen on carbon with the production of carbon monoxide which is dissolved in the iron—we may even have the formation of iron carbonyl. We know that when liquid iron cools, graphite forms together with the eutectic. Any excess of oxygen above 0.010 per cent. would cause an ebullition of gas, because the solubility point of carbon monoxide in iron would be overreached and there would ensue the steel-making reaction:

$$FeO + Fe_3C = 4Fe + CO.$$

It can easily be seen that the percentage of oxygen must be exactly that called for by the physical-chemical effect, and it is a great credit to Mr. Johnson that he has learned this by empirical methods. His deduction from *a priori* reasoning, based on premises calculated from many isolated phenomena, is a brilliant intellectual feat, as is his experimental proof.

It may be that further work on the effect of oxygen in cast iron will lead to the conclusion that a certain percentage of oxygen will improve steel. Possibly reagents other than oxygen might be used. As developed in this paper, the application of a chemical theory to the question of the size of particles of the several components of cast iron and steel has attractive possibilities in the domain of both pure and applied science. Unquestionably the presence of the right amount of a solvent will increase the density of a component, provided conditions are such that the law operates within proper limits.

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